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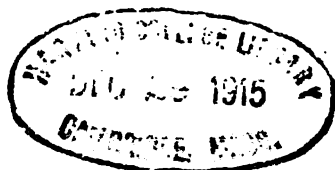
GOGEBIC-CUYUNA RANGES

SEPTEMBER 6, 7, 8, 9, 1915

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IRONMINE, MICH.

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For the year ending with the close of the annual meeting, September 7, 1915.

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A. J. YUNGBLUTH, Secretary	Ishpeming, Mich.

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M. B. M'GEE	Crystal Falls, Mich.
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MINING METHODS ON THE GOGEBIC RANGE, 1915.

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ADGATE, FREDERICK W.	300 Ohio Ave., Warwood, W. Va.
AISHTON, R. H.	215 W. Jackson Blvd., Chicago, Ills.
ALLEN, R. C.	Lansing, Mich.
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 DICKERMAN, ALTON L..... 70 State St., Boston, Mass.
 DIEHL, ALFRED S..... Coleraine, Minn.
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 DUDLEY, HARRY C..... 807 Lonsdale Bldg., Duluth, Minn.
 DUNCAN, MURRAY M..... Ishpeming, Mich.
 DUNSTER, CARL B..... Pine St., New York City, N. Y.

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GARDNER, W. A.....	215 Jackson Blvd., Chicago, Ills.
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GIBSON, WILLIAM M.....	Calumet, Mich.
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HOLTHOFF, HENRY C.....	459 Juneau Place, Milwaukee, Wis.
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IMHOFF, WALLACE G.....	2212 W. 11th St., Cleveland, Ohio
IRELAND, JAMES D.....	701 Sellwood Bldg., Duluth, Minn.
JACKA, JOSIAH S.....	Crystal Falls, Mich.
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JACKSON, GEORGE R..... Princeton, Mich.
 JAMES, D. G..... 312 W. 5th St., Ottumwa, Iowa
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 LONGYEAR, J. M. JR. 406 N. Pinckney St., Madison, Wis.
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M'CARTHY, EDW. P.	Minnesota School of Mines, Minneapolis, Minn.
M'CLURE, O. D.	Ishpeming, Mich.
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M'DONALD, D. B.	303 Glencoe Bldg., Duluth, Minn.
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 OLCOTT, WILLIAM J.....Duluth, Minn.
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PAINE, W. A.....82 Devonshire St., Boston, Mass.
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COWLING, NICHOLAS ...1910	MILLER, A. M.1912
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KRUSE, JOHN C.1907	WILLIAMS, W. H.1897

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COLE, WM. H.	March 8, 1915
SHERLOCK, THOMAS	August 7, 1915
SHERRERD, JOHN M.	April 15, 1915
TRAVER, WILBUR H.	April 15, 1915
*WINCHELL, N. H.	April 2, 1914

*Not reported in 1914.

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1	Iron Mountain, Mich.....	March 22-23, 1893.....	Vol. I
2	Houghton, Mich..	March 7-9, 1894	Vol. II
3	Mesabi and Vermilion Ranges.	March 6-8, 1895.....	Vol. III
4	Ishpeming, Mich..	August 18-20, 1896.....	Vol. IV
5	Ironwood, Mich.....	August 16-18, 1898.....	Vol. V
6	Iron Mountain, Mich.....	February 6-8, 1900.....	Vol. VI
7	Houghton, Mich..	March 5-9, 1901.....	Vol. VII
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11	Iron Mountain, Mich.....	October 17-19, 1905....	Vol. XI
12	Houghton, Mich..	August 8-10, 1906.....	Vol. XII
13	Mesabi and Vermilion Ranges.	June 24-27, 1908	Vol. XIII
14	Ishpeming, Mich..	August 25-27, 1909....	Vol. XIV
15	Ironwood, Mich..	August 24-26, 1910....	Vol. XV
16	Crystal Falls, Mich.....	August 22-24, 1911.....	Vol. XVI
17	Houghton, Mich.....	August 28-30, 1912....	Vol. XVII
18	Mesabi Range	August 26-30, 1913.....	Vol. XVIII
19	Marquette Range..	Aug. 31 to Sept. 3, 1914	Vol. XIX
20	Gogebic-Cuyuna Ranges	Sept. 6 to 9, 1915.....	Vol. XX

Note—No meetings were held in 1897, 1899 and 1907.

RULES OF THE INSTITUTE.

I.

OBJECTS.

The objects of the Lake Superior Mining Institute are to promote the arts and sciences connected with the economical production of the useful minerals and metals in the Lake Superior region, and the welfare of those employed in these industries, by means of meetings of social intercourse, by excursions, and by the reading and discussion of practical and professional papers, and to circulate, by means of publications among its members, the information thus obtained.

II.

MEMBERSHIP.

Any person interested in the objects of the Institute is eligible for membership.

Honorary members not exceeding ten in number, may be admitted to all the privileges of regular members except to vote. They must be persons eminent in mining or sciences relating thereto.

III.

ELECTION OF MEMBERS.

Each person desirous of becoming a member shall be proposed by at least three members, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe), upon receiving three-fourths of the votes cast. Application must be accompanied by fee and dues as provided by Section V.

Each person proposed as an honorary member shall be recommended by at least ten members, approved by the Council, and elected by ballot at a regular meeting, (or by ballot at any time conducted through the mail, as the Council may prescribe), on receiving nine-tenths of the votes cast.

IV.

WITHDRAWAL FROM MEMBERSHIP.

Upon the recommendation of the Council, any member may be stricken from the list and denied the privilege of membership, by the vote of three-fourths of the members present at any regular

meeting, due notice having been mailed in writing by the Secretary to him.

V.

DUES.

The membership fee shall be five dollars and the annual dues five dollars, and applications for membership must be accompanied by a remittance of ten dollars; five dollars for such membership fee and five dollars for dues for the first year. Honorary members shall not be liable to dues. Any member not in arrears may become a life member by the payment of fifty dollars at one time, and shall not be liable thereafter to annual dues. Any member in arrears may, at the discretion of the Council, be deprived of the receipt of publications or be stricken from the list of members when in arrears six months; Provided, That he may be restored to membership by the Council on the payment of all arrears, or by re-election after an interval of three years.

VI.

OFFICERS.

There shall be a President, five Vice Presidents, five Managers, a Secretary and a Treasurer, and these Officers shall constitute the Council.

VII.

TERM OF OFFICE.

The President, Secretary and Treasurer shall be elected for one year, and the Vice Presidents and Managers for two years, except that at the first election two Vice Presidents and three Managers shall be elected for only one year. No President, Vice President, or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. The term of office shall continue until the adjournment of the meeting at which their successors are elected.

Vacancies in the Council, whether by death, resignation, or the failure for one year to attend the Council meetings, or to perform the duties of the office, shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; Provided, That such appointment shall not render him ineligible at the next election.

VIII.

DUTIES OF OFFICERS.

All the affairs of the Institute shall be managed by the Council except the selection of the place of holding regular meetings.

The duties of all Officers shall be such as usually pertain to their offices, or may be delegated to them by the Council.

The Council may, in its discretion, require bonds to be given by the Treasurer, and may allow the Secretary such compensation for his services as they deem proper.

At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Five members of the Council shall constitute a quorum; but the Council may appoint an executive committee, business may be transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to the approval of a majority of the Council, subsequently given in writing to the Secretary and recorded by him with the minutes.

There shall be a meeting of the Council at every regular meeting of the Institute and at such other times as they determine.

IX.

ELECTION OF OFFICERS.

Any five members not in arrears, may nominate and present to the Secretary over their signatures, at least thirty days before the annual meeting, the names of such candidates as they may select for offices falling under the rules. The Council, or a committee thereof duly authorized for the purpose, may also make similar nominations. The assent of the nominees shall have been secured in all cases.

No less than two weeks prior to the annual meeting, the Secretary shall mail to all members not in arrears a list of all nominations made and the number of officers to be voted for in the form of a letter ballot. Each member may vote either by striking from or adding to the names upon the list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing the ballot with his name, and either mailing it to the Secretary, or presenting it in person at the annual meeting.

In case nominations are not made thirty days prior to the date of the annual meeting for all the offices becoming vacant under the rules, nominations for such offices may be made at the said meeting by five members not in arrears, and an election held by a written or printed ballot.

The ballots in either case shall be received and examined by three tellers appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected. The ballot shall be destroyed, and a list of the elected officers, certified by the tellers, shall be preserved by the Secretary.

X.

MEETINGS.

The annual meeting of the Institute shall be held at such time as may be designated by the Council. The Institute may at a regular

meeting select the place for holding the next regular meeting. If no place is selected by the Institute it shall be done by the Council.

Special meetings may be called whenever the Council may see fit; and the Secretary shall call a special meeting at the written request of twenty or more members. No other business shall be transacted at a special meeting than that for which it was called.

Notices of all meetings shall be mailed to all members at least thirty days in advance, with a statement of the business to be transacted, papers to be read, topics for discussion and excursions proposed.

No vote shall be taken at any meeting on any question not pertaining to the business of conducting the Institute.

Every question that shall properly come before any meeting of the Institute, shall be decided, unless otherwise provided for in these rules, by the votes of a majority of the members then present.

Any member may introduce a stranger to any regular meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

XI.

PAPERS AND PUBLICATIONS.

Any member may read a paper at any regular meeting of the Institute, provided the same shall have been submitted to and approved by the Council, or a committee duly authorized by it for that purpose prior to such meeting. All papers shall become the property of the Institute on their acceptance, and with the discussion thereon, shall subsequently be published for distribution. The number, form and distribution of all publications shall be under the control of the Council.

The Institute is not, as a body, responsible for the statements of facts or opinion advanced in papers or discussions at its meetings, and it is understood, that papers and discussions should not include personalities, or matters relating to politics, or purely to trade.

XII.

SPECIAL COMMITTEES.

The Council is authorized to appoint from time to time special committees to consider and report upon, to the Institute through the Council, such subjects as changes in mining laws, safety devices, the securing and editing of papers on mining methods, definition of mining terms, affiliations with other societies, and such other subjects as the Council shall deem it desirable to inquire into, such reports not to be binding on the Institute except action is taken by the Institute in accordance with the rules, and the Council is au-

thorized to expend not exceeding six hundred dollars in any one year to carry out the purpose of this section.

XIII.

AMENDMENTS.

These rules may be amended by a two-thirds vote taken by letter ballot in the same manner as is provided for the election of officers by letter ballot; Provided, That written notice of the proposed amendment shall have been given at a previous meeting.

PROCEEDINGS OF THE TWENTIETH ANNUAL MEETING, GOGEBIC RANGE.

MONDAY, SEPTEMBER 6, 1915.

The Twentieth Annual Meeting of the Institute opened on the Gogebic Range, (Mich.) with headquarters at Ironwood. There were about two hundred members and guests in attendance. At 9:30 the party proceeded in automobiles to the ball park where the First-Aid demonstration was held. Fourteen teams from the different iron ranges participated in the contest, and the attendance numbered over two thousand. It being Labor Day a number of the miners with their families witnessed the exhibition, and manifested great interest in the various events. This was the second contest held in the Lake Superior district under the auspices of the Institute. A full account with the events, rules of the contest, and other information is published in a special chapter in this volume.

In the afternoon the visitors were conveyed in automobiles to the mines east of Ironwood, and the new operations at the extreme eastern end of the range where the Wakefield Mine is being opened. This mine is fully described in a paper by W. C. Hart, Superintendent of the property. The plant at the Woodbury Shaft of the Newport Mining Company was also visited. The record of sinking this shaft as described in the paper by J. M. Broan, was of much interest to the mining men. The "one-leg" wood-stocking trestle at the Colby Mine attracted much attention. This is described in the paper by G. S. Barber, Superintendent of the Colby Mining Company. The surface plants of the Pabst, Anvil, Palms and Norrie Mines were also visited during the trip.

BUSINESS SESSION.

The business session in the evening was held at the new

Central High School. The meeting was called to order by L. M. Hardenburgh, President, who made a brief address of welcome on behalf of the members from the Gogebic Range upon this, the fourth meeting to be held on this range. It was in Ironwood in 1893, that the Institute had its inception and the progress of the development of its mines has been fully recorded in its proceedings since that time.

The following papers were presented in oral abstract by the authors:

*Sinking of the Woodbury Shaft at the Newport Mine, Ironwood, Mich.—By J. M. Broan, Ironwood, Mich.

*Mining Methods on the Gogebic Range—Report of Committee, presented by O. E. Olsen, Ironwood, Mich.

*New Stockpile Trestle, Colby Iron Mining Company, Bessemer, Mich.—By G. S. Barber, Bessemer, Mich.

*The Opening of the Wakefield Mine—By W. C. Hart, Wakefield, Mich.

*Grouting at the Francis Mine Shaft of The Cleveland-Cliffs Iron Co.—By J. R. Reigart, Princeton, Mich.

*Sheet Ground Mining in the Jopling District, Missouri—By Edwin Higgins, Ironwood, Mich.

*The Use of Gunite in a Steel Shaft and in an Underground Pumphouse, on the Gogebic Range—By Stephen Royce, Hurley, Wis.

This concluded the reading of papers for the session.

On motion by J. M. Bush, the President appointed the following committee on nominations: J. M. Bush, G. L. Woodworth, W. J. Richards, Crystals Falls, W. P. Chinn, and F. W. Denton.

On motion by C. H. Baxter, the President appointed the following committee to audit the accounts of the Secretary and Treasurer: C. H. Baxter, F. B. Goodman and J. E. Jopling.

On motion by William Kelly, the President appointed the following committee on resolutions: William Kelly, L. C. Brewer, J. H. Harding, J. Carroll Barr and Chas. L. Lawton.

Committees to report at the next session to be held Tuesday afternoon at Crosby, Minnesota. After making the an-

*Papers distributed in printed form.

nouncements of the program for the next day, the meeting was on motion adjourned.

At 11:00 o'clock p. m., the party left by special train, consisting of twenty coaches, over the Soo Line for Crosby, Minnesota.

TUESDAY, SEPTEMBER 7TH.

Tuesday morning at 9:00 o'clock the party arrived at Crosby, Minnesota, from which point the inspection of the Cuyuna Range was commenced. A train of flat cars provided with benches and railings, was furnished for a trip over the range, and afforded a splendid opportunity for a close inspection of the mines. The first stop was made at the Kennedy Mine of the Rogers-Brown Ore Company at Cuyuna. This is an underground mine and the pioneer on the range. It was opened in 1907, and made its first shipment in 1911 of 147,431 tons. The next stop was at the Croft Mine of the Merrimac Mining Company, near the town of Crosby. A circular concrete shaft was sunk to the ledge, and sinking is now going on in the rock. This is said to be the first mine which will average a bessemer grade of ore.

The Meacham Mine, also the property of the Rogers-Brown Ore Company, was next visited. A circular concrete shaft was sunk to the ledge. The shaft is down to the ore-body, a depth of 235 ft., and cross-cutting started to the ore to the south. The property is temporarily idle.

The Thompson, Armour No. 2 and Armour No. 1, were each visited in the order named. These mines are operated by the Inland Steel Company. The Thompson was first opened in 1911, and Armour No. 1 and No. 2 in 1910. A small washing plant is being operated at the Thompson to treat the siliceous ores.

The Pennington property, operated by the Tod-Stambaugh Company, was the next visited. This is an open-pit mine and has the distinction of being the first stripping operation on the range. It is expected that a considerable tonnage will be shipped before the close of the present season.

The next stop was at the Hillcrest Mine of the Hill Mines Company. Here the party witnessed the hydraulic stripping method which is being developed to a very satisfactory degree in point of economy. The surface averages about 65 ft. in depth. Steam shovels will be necessary to clean up the bottom of the pit after hydraulic stripping is completed to uncover the ore. The plant is operated electrically by power furnished from the Cuyuna Range Power Company.

The next property to be inspected was the Rowe Mine of the Pittsburgh Steel Ore Company. This is the most westerly of the developed properties on the north range and the largest open-pit operation in the district and said to be the first to use the hydraulic method of stripping. The special train was transferred to the ore company's crews and run into the open pit where steam-shovel stripping is still going on. After an inspection of the mine the party was taken to the Club House, a very pretty building of the bungalow type, situated on the hill overlooking the mine and an arm of the Mississippi River. Here luncheon was served to the two hundred visitors in quick time, Mr. Barr acting as master of ceremonies. Immediately after the luncheon the concentrating plant for washing the lean ores was inspected.

The party then returned to Crosby where the business session was held at the Franklin school house.

Time did not permit a visit to the South Range and the City of Brainerd and the committee planned that the district would be again visited some time in the near future.

BUSINESS SESSION.

In the absence of Mr. Hardenburgh, L. C. Brewer, Vice-President, presided at the meeting, which was called to order at 3:45 p. m. The following papers were presented in oral abstract:

*Some Aspects of Explorations and Drilling on the Cuyuna Range—By P. W. Donovan, Brainerd, Minn., presented by Carl Zapffe.

Interesting Matters to Operators Regarding Cuyuna District—By Carl Zapffe, Brainerd, Minn.

*A Survey of the Developments and Operations in the Cuyuna Iron Ore District of Minnesota—By Carl Zapffe, Brainerd, Minn.

Concentration of Cuyuna Ores—By Edmund Newton, Minnesota School of Mines, Minneapolis, Minn., presented by E. H. Comstock.

Hydraulic Stripping at the Rowe and Hillcrest Mines—By Edward P. McCarthy, Minnesota School of Mines, Minneapolis, Minn.

The following papers were read by title:

*Rock Drifting in the Morris-Lloyd Mine, Marquette Range—By J. Ellzey Hayden, Ishpeming, Mich.

*The Mining School of The Cleveland-Cliffs Iron Company—By C. S. Stevenson, Ishpeming, Mich.

Progress in Underground Mechanical Ore Loading—By M. Earl Richards, Crystal Falls, Mich.

Drag-Line Stripping and Mining, Balkan Mine—By Chas. E. Lawrence, Palatka, Mich.

This concluded the presentation of papers for the session.

*Papers distributed in printed form.

REPORT OF THE COUNCIL.

Secretary's report of Receipts and Disbursements from August 24, 1914, to August 31, 1915.

Receipts.

Cash on hand, August 24, 1914.....		\$6,823 65
Entrance fees for 1914	\$ 180 00	
Dues for 1914	2,080 00	
Back dues, 1910	\$ 5 00	
Back dues, 1911	10 00	
Back dues, 1912	20 00	
Back dues, 1913	70 00	105 00
Advance dues, 1915	40 00	
Sale of Proceedings	77 35	
Institute Pin	4 00	
Total		\$2,486 35
*Interest on deposits		169 07
Total receipts		2,655 42
Grand total on hand and received..		\$9,479 07

*Interest on bonds earned but not due \$100.00

Disbursements.

Stationery and printing	\$ 63 50	
Postage	133 66	
Freight and express	6 03	
Exchange	1 90	
Telephone and telegraphing	3 37	
Secretary's salary	750 00	
Stenographic work	89 90	
Total office expenses		\$1,048 36
Publishing Proceedings	\$1,131 50	
Publishing advance papers	298 75	
Photographs, maps, cuts, etc.....	80 68	
Badges for 1914	81 25	
Expenses of committee meetings.....	35 14	
Donation to First-Aid Contest.....	50 00	
Total		1,677 32
Total disbursements		2,725 68
Cash on hand, August 31, 1915.....		6,753 39
Grand total		\$9,479 07

Membership.

	1915.	1914.	1913.
Total	549	549	518
Members in good standing	*501	1524	483
Honorary members	3	4	3
Life members	2	2	2
Members in arrears (2 years).....	43	19	29
New members admitted	30	36	71
New members not qualified	5
New members added	30	36	66

*Includes 77 in arrears for one year. Includes 54 in arrears for one year.

TREASURER'S REPORT.

Treasurer's Report from August 24, 1914, to August 31, 1915:

Cash on hand, August 24, 1914.....	\$6,823.65	
Received from Secretary	2,486.35	
Received interest on deposits	169.07	
Paid drafts issued by Secretary		\$2,725.68
Cash on hand, August 31, 1915		6,753.39
Totals	\$9,479.07	\$9,479.07

On authority of the Council, by vote taken by letter ballot, the Treasurer reported the purchase of Alpha Water Works bonds in the sum of \$5,000.00. These bonds are issued by the Village of Alpha, Iron County, Michigan, and mature in twelve years; interest at six per cent.

The following standing committees were appointed by the Council for the ensuing year:

"PRACTICE FOR THE PREVENTION OF ACCIDENTS."

(Committee to consist of five members).

William Conibear, Ishpeming, Mich., Chairman; Percy S. Williams, Ramsay, Mich.; William H. Jobe, Crystal Falls, Mich.; Elton W. Walker, Mass, Mich.; W. H. Harvey, Eveleth, Minn.

"CARE AND HANDLING OF HOISTING ROPES."

(Committee to consist of five members).

William J. Richards, Painesdale, Mich., Chairman; Joseph Kieren, Gilbert, Minn.; Frank H. Armstrong, Vulcan, Mich.; Carlos E. Holley, Bessemer, Mich.; C. M. Murphy, Ishpeming, Mich.

"PAPERS AND PUBLICATIONS."

(Committee to consist of five members).

William Kelly, Vulcan, Mich., Chairman; Frederick W. McNair, Houghton, Mich.; James E. Jopling, Ishpeming, Mich.; Frank Blackwell, Ironwood, Mich.; Alexander M. Gow, Duluth, Minn.

"BUREAU OF MINES."

(Committee to consist of three members).

Murray M. Duncan, Ishpeming, Mich., Chairman; Frederick W. Denton, Painesdale, Mich.; A. J. Yungbluth, Ishpeming, Mich., Secretary.

"BIOGRAPHY."

(Committee to consist of five members).

John H. Harding, Duluth, Minn., Chairman; Robert A. Douglas, Ironwood, Mich.; M. B. McGee, Crystal Falls, Mich.; W. H. Newett, Ishpeming, Mich.; James Fisher, Houghton, Mich.

"MINING METHODS ON THE GOGEBIC RANGE."

(Committee to consist of three members to be appointed later).

Committees to serve until their successors are appointed; each committee to have power to appoint sub-committees as may be deemed necessary.

The following proposals for membership are approved by the Council:

Broan, J. M., Mining Engineer, Newport Mining Co., Ironwood, Mich.

Carlson, Gust, Diamond Drill Contractor, Hibbing, Minn.

Cardle, James, President, Mutual Iron Mining Co., Duluth, Minn.

Collins, Chas. D., Physician, Newport Iron Mining Co., Ironwood, Mich.

Collins, Edwin J., Mining Engineer, (Consulting) Torrey Bldg., Duluth, Minn.

Constable, William, Salesman, General Electric Co., 801 Fidelity Bldg., Duluth, Minn.

Cullen, E. L., Manager, Newport Mining Co., Ironwood, Mich.

Erickson, Gustaf A., Mining Captain, Oliver Iron Mining Co., Ironwood, Mich.

Hansen, Christ, Superintendent, Board of Public Works, Negaunee, Mich.

Hanson, W. G., Superintendent, Palatka, Mich.

Hill, Edmund, Mining Captain, Newport Mining Co., Ironwood, Mich.

Hotchkiss, William O., State Geologist of Wisconsin, Madison, Wis.

James, D. G., Salesman, Ottumwa Iron Works, 312 W. 5th St., Ottumwa, Iowa.

Johnson, John A., Mining Captain, Wakefield Mine, Wakefield, Mich.

Kruka, Erick W., Chief Clerk, Champion Copper Co., Painesdale, Mich.

Kyler, E. R., Mechanical Engineer, Commonwealth, Wis.

Lawry, Henry M., Mining Captain, Palatka, Mich.

Longyear, John M. Jr., Mining Engineer and Geologist, 406 N. Pinckney St., Madison, Wis.

Madson, Jesse C., Mining Captain, Carson Lake, Minn.

McCarty, Edward P., Professor of Mining, Minnesota School of Mines, Minneapolis, Minn.

McKenna, Edward B., Salesman, Adolph Hirsch & Co., Duluth, Minn.

Olsen, Oscar E., Mining Engineer, Oliver Iron Mining Co., 403 N. Lawrence St., Ironwood, Mich.

Pearl, Holman I., Mining Engineer, Wakefield, Mich.

Perkirs, William J., Mine Superintendent, Alpha Iron Co., Alpha, Mich.

Roberts, H. M., Geologist, 710 Security Bank Bldg., Minneapolis, Minn.

Rossman, Lawrence A., Mining Editor, Herald-Review, Grand Rapids, Mich.

Schenck, Charles H., Salesman, The United States Graphite Co., 2624 Lyndale Ave., So. Minneapolis, Minn.

Scott, Thaddeus, Secretary, Mutual Iron Mining Co., 518 Providence Bldg., Duluth, Minn.

Truettner, Walter F., Banker, Bessemer, Mich.

Wildes, F. A., Chief Inspector of Mines for Auditor of State of Minnesota, Hibbing, Minn.

On motion by F. W. McNair, the Secretary was instructed to cast a ballot for the election to membership of the list as approved by the Council.

The Auditing Committee presented the following report:

Your Committee appointed to examine the books of the Secretary and Treasurer, beg leave to report that we have carefully examined same and find the receipts and expendi-

tures shown therein to be in accordance with the statements of the Secretary and Treasurer for the fiscal year ending August 31, 1915.

FRANK B. GOODMAN,
J. E. JOPLING,
C. H. BAXTER,
Committee.

REPORT OF COMMITTEE ON NOMINATION.

Your Committee on Nominations beg leave to submit the following Officers of the Institute for terms specified:

For President (one year)—Charles E. Lawrence.

For Vice Presidents (two years)—George L. Woodworth, Frank E. Keese, Grant S. Barber.

For Managers (two years)—Frank Armstrong, William Wearne.

For Treasurer (one year)—E. W. Hopkins.

For Secretary (one year)—A. J. Yungbluth.

J. M. BUSH,
F. W. DENTON,
G. L. WOODWORTH,
W. P. CHINN,
W. J. RICHARDS,
Committee.

On motion the report of the Committee was adopted and the Secretary instructed to cast a ballot for the election of the officers for the terms specified.

The following communications were read:

To the President of

Lake Superior Mining Institute.

Dear Sir:

By virtue of the authority conferred upon me by the Congress of the United States of America, I have the pleasure to extend to Lake Superior Mining Institute a cordial invitation to participate by one or more delegates in The Second Pan-American Scientific Congress to be held under the auspices of the Government of the United States at the City of Washington from December 27, 1915, to January 8, 1916, inclusive.

Assuring you that representatives from the Institute will be most heartily welcomed,

I am, my dear Sir,

Very truly yours,

W. J. BRYAN,

Department of State.

Secretary of State.

Washington, February 12, 1915.

On motion duly seconded, it was decided that the Institute would not send a delegate to this Congress.

A. J. Yungbluth, Esq.,

Sec'y., Lake Superior Mining Institute,
Ishpeming, Mich.

Dear Sir:

The President and Councillors of the Mining and Metallurgical Society of America invite the Lake Superior Mining Institute to be represented by delegates at a meeting of the Mining and Metallurgical Society of America, to be held in Washington, D. C., on Thursday, December 16th, 1915; they also extend a general invitation to, and request the attendance of all of your members who are interested in the objects of the gathering.

The purpose of this meeting is to bring before the members of Congress and other Washington officials, facts and arguments bearing on the necessity of certain changes in the mining laws of the United States.

In the spring of 1914, bills were introduced in both houses recommending the appointment of a commission to take testimony, codify and to suggest amendments to the general mining laws. The Senate's committee on Mines and Mining recommended for passage, with certain modifications, the Smoot bill (S. 4373). The House committee on Mines and Mining, in the same way, recommended for passage the Taylor bill (H. R. 15283). Both failed of passage, mainly because of the pressure of other matters.

The Mining and Metallurgical Society, having carefully canvassed the opinion of its members, believes that there are many points requiring alteration, upon which all those engaged in mining are practically unanimous in their views, and we believe that, with the co-operation of other organizations interested in the same subject, sufficient pressure may be brought to bear to produce the results, accomplishment of which has failed in the past.

All of the public officials at Washington, with whom we have consulted, are in accord with our views and have promised their support and assistance in placing our requirements before Congress and the Senate in such shape and with such force as to secure some action.

To accomplish the desired results we wish to secure a large and representative attendance of those who may speak with authority on the desirability and necessity of such alterations as should be considered, and we trust that the directors of your organization may see their way to co-operate with us in this undertaking.

The Secretary of the Mining and Metallurgical Society would be pleased to receive, at an early date, the names of those whom your Society may see fit to appoint as delegates.

Enclosed herewith you will find a copy of the last progress report of our committee on mining law, which indicates briefly the present status of the matter.

Yours very truly,

F. F. SHARPLESS, Secretary.

On motion duly seconded, the communication was referred back to the Council with power to act.

The communication from the committee on "Practice for the Prevention of Accidents" relative to defraying the expenses of the winning team in the First-Aid contest to the Panama-Pacific Expositions, was submitted to the council. In view of the short time remaining before this event took place the council decided it was beyond its authority to take any action in the matter and that it would be necessary to lay the question before the Institute at a future meeting.

Following the close of the meeting the party proceeded to the farm of George H. Crosby, where a barbecue was tendered the visitors, after which an entertainment was provided on the shore of Serpent Lake. Music was furnished by the Crosby orchestra. Mr. Crosby welcomed the guests and gave a brief address on the early days of the village. This was followed by songs and speeches from several of the visitors. A huge bon-fire furnished the illumination for the occasion.

At 10:30 p. m., the party left by two special trains via Soo Line for Minneapolis, where the meeting came to a close, arriving there at 9:00 o'clock, Wednesday.

WEDNESDAY, SEPTEMBER 8TH.

The Minneapolis Civic and Commerce Association very kindly arranged for the entertainment of the party at Minneapolis. The following citizens constituted the committee on this occasion: Chas. H. Robinson, Frank W. Plant, Fred B. Snyder, C. S. Langdon, Russell M. Bennett, John Pillsbury, J. C. Van Doorn, J. P. Snyder, Frank Bovey, H. V. Winchell, F. G. Jewett, Jos. Chapman, Geo. H. Warren, J. R. Vanderlip, Carl DeLaittre, W. L. Martin.

Automobiles were provided to convey the party about the city, and the trip through the various parks was a very enjoyable experience for the members. The lakes, in which the locality abounds, add much to the picturesqueness of the drive. Upon arriving at the Minnesota School of Mines the party was met by Dean Appleby and members of the faculty, and was shown through the new building then nearing completion. The furnishings are all new and substantial, and the equipment of the latest and best. Considerable time was spent here and the visit was much enjoyed by all.

From here the party proceeded to the Minikahda Club, where a substantial luncheon was served and the visitors royally entertained. Those desiring to spend the afternoon in golf were accommodated at various country clubs, while many others visited the State Fair.

Many of the visitors remained in the city for the balance of the week, all voting the trip to Minneapolis a very enjoyable feature.

The following is the report submitted by the Committee on Resolutions:

Resolved, by the members of the Lake Superior Mining Institute in attendance at the 1915 meeting, that we hereby extend our thanks to the Gogebic Range Mining Association, the Norrie and Newport bands, the owners of automobiles, and all others of the district who contributed to the entertainment of the Institute members and made the stay on the Gogebic Range a very pleasant one, and

Also, to the Du Pont Powder Company and the business men of Ironwood, for prizes which were donated for the First-

Aid contest, and to Dr. A. E. Knoefel, Edwin Higgins, and the others, who officiated so ably at the exhibition, and to those who made it possible for the teams to participate, and

Also, to the officials of the Minneapolis, St. Paul & Sault Ste. Marie Railway Co., who provided the excellent train service and extended a number of courtesies, which were highly appreciated, and

Also, to J. C. Barr, the Pittsburgh Steel Ore Company, George H. Crosby, and the Commercial Club of Crosby, who afforded such splendid entertainment on the Cuyuna Range, and

Also, to the Minneapolis Civic and Commercial Club, which acted as host in Minneapolis, and to President Vincent, Dean Appleby, and other officials of the University of Minnesota, who cordially showed us the Mining Department of that great institution, and

Also, to the authors who kindly responded with papers for this meeting.

WILLIAM KELLY,
L. C. BREWER,
J. H. HEARDING,
J. CARROLL BARR,
CHAS. L. LAWTON,
Committee.

The following is a partial list of those in attendance:

Abeel, G. H....Ironwood, Mich.	Champion, Chas..Beacon, Mich.
Andrews, C. E..Escanaba, Mich.	Chinn, W. P.....Gilbert, Minn.
	Chisholm, A. D..Bessemer, Mich.
Barber, G. S...Bessemer, Mich.	Clifford, J. M..Green Bay, Wis.
Barrows, W. A. Jr..Brainerd, Minn.	Cole, C. D....Ishpeming, Mich.
Baxter, C. H....Loretto, Mich.	Cole, W. A....Ironwood, Mich.
Bengry, W. H...Palatka, Mich.	Collins, C. D...Ironwood, Mich.
Berteling, J. F..Ishpeming, Mich.	Comstock, E. H.....
Blackwell, F....Ironwood, Mich. Minneapolis, Minn.
Bolles, F. R....Houghton, Mich.	Conibear, W....Ishpeming, Mich.
Bond, Wm....Ironwood, Mich.	Connors, Thos..Negaunee, Mich.
Brewer, L. C...Ironwood, Mich.	Cory, Edwin...Negaunee, Mich.
Broan, J. M....Ironwood, Mich.	Crosby, G. H....Duluth, Minn.
Broan, JohnChicago, Ills.	
Burdorf, H. A..Minneapolis, Minn.	Davis, W. J....Wakefield, Mich.
Burnham, L. W..St. Paul, Minn.	Dean, Dudley S..Boston, Mass.
Bush, J. M., , , , Republic, Mich.	Denton, F. W..Painesdale, Mich.

- Dickerson, L. R...Chicago, Ills.
 Douglas, R. A...Ironwood, Mich.
 Edwards, A. D...Atlantic, Mich.
 Eldredge, P. C...Milwaukee, Wis.
 Ericson, G.....Ironwood, Mich.
 Ericson, Gustaf.Ironwood, Mich.
 Fairbairn, C.T..Birmingham, Ala.
 Fay, Joseph...Marquette, Mich.
 Fisher, James..Houghton, Mich.
 Flodin, N. P...Marquette, Mich.
 Gardner, O. D..Houghton, Mich.
 Goodman, F. B...Hurley, Wis.
 Goodney, S.J...Stambaugh, Mich.
 Gowling, T.A...Marquette, Mich.
 Gribble, S. J...Ironwood, Mich.
 Hallingby, Ole...Calumet, Mich.
 Hanson, W. G...Palatka, Mich.
 Hanson, C.....Negaunee, Mich.
 Hardbenburgh, L.M...Hurley, Wis
 Hathaway, G...Ishpeming, Mich.
 Hayden, J. E...Ishpeming, Mich.
 Hearing, J. H...Duluth, Minn.
 Helmer, C. E...Escanaba, Mich.
 Hickok, D. R.....Antigo, Wis.
 Higgins, Edwin..Pittsburgh, Pa.
 Hildreth, T. F...Buffalo, N. Y.
 Hill, Edmund ..Ironwood, Mich.
 Hoatson, Thos...Laurium, Mich.
 Holman, J. W.....Chicago, Ills.
 Hopkins, E. W.....
 Commonwealth, Wis.
 Hoskins, Samuel...Hurley, Wis.
 Hunner, E. E.....Duluth, Minn.
 Ireland, J. D.....Duluth, Minn.
 Ives, L. E.....New York, N. Y.
 Jackson, G. R..Princeton, Mich.
 Johnson, H. O...Virginia, Minn.
 Johnson, J. A...Wakefield, Mich.
 Johnstone, O. W...Duluth, Minn.
 Jolly, John....Painesdale, Mich.
 Jopling, J. E...Ishpeming, Mich.
 Kates, C. W.....Wells, Mich.
 Keast, George....Norway, Mich.
 Kelly, William....Vulcan, Mich.
 King, Robert...Ironwood, Mich.
 Kirkpatrick, J. C. Jr.....
 Park Falls, Wis.
 Knight, J. B....Norway. Mich.
 Kruka, E. W..Painesdale, Mich.
 LaRochelle, L..Houghton, Mich.
 LaRue, W. G....Duluth, Minn.
 Lawry, H. M....Palatka, Mich.
 Lawton, C. L...Hancock, Mich.
 Lesselyong, F.H..Ironwood, Mich
 Letz, John F...Milwaukee, Wis.
 Lukey, Frank.....Hurley, Wis.
 Lukey, F. G....Houghton, Mich.
 Lutes, J. S.....Biwabik, Minn.
 Lytle, C. E....Marquette, Mich.
 Madson, J.C..Carson Lake, Minn.
 Martin, Al...Crystal Falls, Mich.
 Matthews, C. H...Duluth, Minn.
 Mitchell, H. E...Eveleth, Minn.
 Moore, W. H...Ironwood, Mich.
 Morgan, D. T....Detroit, Mich.
 McDonald, D. B..Duluth, Minn.
 McNair, F. W..Houghton, Mich.
 McNamara, T.B...Ironwood, Mich.
 McRandle, W.E...Bessemer, Mich.
 Nelson, J. E...Negaunee, Mich.
 Newett, W. H...Ishpeming, Mich.
 Noetzel, B.D..Trimountain, Mich
 Olsen, O. E....Ironwood, Mich.
 Pascoe, P. W...Republic, Mich.
 Pearce, E. L..Marquette, Mich.
 Pearl, H. I....Wakefield, Mich.
 Prescott, F.M...Menominee, Mich.
 Quigley, G. J.....Antigo, Wis.
 Quine, J. T....Ishpeming, Mich.
 Quinn, J. H...Ishpeming, Mich.

- Raisky, F. H. Duluth, Minn.
 Reigart, J. R. Princeton, Mich.
 Richards, W. J. Painesdale, Mich.
 Richards, M. E.
 Crystal Falls, Mich.
 Richards, W. J.
 Crystal Falls, Mich.
 Roberts, H. M. Minneapolis, Minn.
 Roberts, A. T. Marquette, Mich.
 Rossman, L. A.
 Grand Rapids, Minn.
 Rough, J. H. Negaunee, Mich.
 Rough, J. H. Jr. Negaunee, Mich.

 Sampson, Jas. Ironwood, Mich.
 Sampson, John Ashland, Wis.
 Sawhill, R. V. Cleveland, Ohio
 Scadden, F. Crystal Falls, Mich.
 Schenck, C. H. Saginaw, Mich.
 Sheldon, A. F. Marquette, Mich.
 Shove, B. W. Ironwood, Mich.
 Shove, Byron. Ironwood, Mich.
 Siebenthal, W. A. Vulcan, Mich.
 Silver, C. R. Chicago, Ills.
 Small, H. H. Chicago, Ills.
 Speare, J. H. Ironwood, Mich.
 Sperr, F. W. Houghton, Mich.
 Sperr, R. Houghton, Mich.
 Soady, Harry Duluth, Minn.
 Stephens, Jas. Ishpeming, Mich.
 Stevenson, C. S. Ishpeming, Mich.
 Stewart, H. E. Houghton, Mich.
 Stoik, G. M. Ironwood, Mich.

 Strachan, W. H. Duluth, Minn.
 Sullivan, J. A. Ironwood, Mich.

 Talboys, H. H. Duluth, Minn.
 Trebilcock, William
 N. Freedom, Wis.
 Truettner, W. F. Bessemer, Mich.
 Trevarthen, W. J.
 Bessemer, Mich.
 Trudgeon, J. Wakefield, Mich.
 Tubby, C. W. St. Paul, Minn.

 VanEvera, W. Virginia, Minn.
 Vivian, G. J. Duluth, Minn.
 Vogel, F. A. New York, N. Y.

 Walker, E. W. Mass, Mich.
 Wallene, F. O. Minneapolis, Minn.
 Ware, W. F. Negaunee, Mich.
 Watson, C. H. Crystal Falls, Mich.
 Wearne, Wm. Hibbing, Mich.
 Webb, W. M. Gilbert, Minn.
 Webb, F. J. Duluth, Minn.
 Webb, C. E. Houghton, Mich.
 Wheeler, H. A. St. Louis, Mo.
 Wildes, F. A. Hibbing, Minn.
 Williams, P. S. Ramsay, Mich.
 Woodworth, G. L. Iron River, Mich.

 Yates, W. H. Negaunee, Mich.
 Yungbluth, A. J. Ishpeming, Mich.

 Zapffe, Carl. Brainerd, Minn.

LOCAL COMMITTEES FOR 1915 MEETING.

Arrangements.

L. C. Brewer, Chairman.

G. S. Barber	R. P. Zinn	F. B. Goodman
W. E. McRandle	G. J. Quigley	O. W. Johnstone
	B. W. Shove	

Finance.

D. E. Sutherland, Chairman.

E. L. Cullen	F. B. Goodman	Robert King
P. S. Williams	William Hart	G. S. Barber
	W. E. McRandle	

Reception.

Henry Rowe, Chairman.

Geo. H. Abeel	C. M. Anderson	C. E. Bennett
William Bond	John Clemens	W. A. Cole
Dr. W. C. Conley	S. S. Cooper	S. S. Curry
George Curry	James Devoy	Geo. O. Driscoll
Con Geary	F. J. Hager	Dr. Hayes Kelly
F. H. Lesselyong	R. McDonald	Dr. Geo. Moore
Oscar Nordling	J. W. Oxnam	R. W. Shand
Jerry Shea	T. J. Stevens	F. J. Sullivan
J. A. Sullivan	F. F. Thalner	Dr. E. H. Madajesky
Dr. J. H. Urquhart	W. J. Zinn	Geo. Lambrix
Henry Meade	Dan Reid	A. L. Ruggles
J. F. Sullivan	Dr. A. Uren	Dr. C. C. Urquhart
Dr. F. G. VanStratum	W. S. Baird	E. R. Bayliss
C. E. Holley	Dr. L. O. Houghton	George McKinney
Dr. W. Pinkerton	W. C. Rowe	W. J. Trevarthen
W. F. Truettner	W. J. Davies	I. W. Truettner
Dr. E. H. Eddy	Ed. Neidhold	Dr. D. C. Pierpont
	Dr. Collins	

Cuyuna Range Committee.

George H. Crosby, Chairman.

J Carroll Barr	J. S. Lutes	Frank Hutchinson
G. A. Anderson	E. J. Donahue	John A. Savage
William Wearn	Wilbur VanEvera	Capt. McGuire
	Carl Zapffe	

Minneapolis Civic and Commerce Association Committee.

Chas. H. Robinson	Fred B. Snyder	C. S. Langdon
Russell M. Bennett	John Pillsbury	J. C. Van Doorn
J. P. Snyder	Frank Bovey	H. V. Winchell
F. G. Jewett	Frank W. Plant	Jos. Chapman
Geo. H. Warren	J. R. Vanderlip	Carl DeLaittre
	W. L. Martin	

GENERAL DESCRIPTION OF THE GOGEBIC RANGE.

BY STEPHEN ROYCE, HURLEY, WIS.*

The iron-bearing formation of the Gogebic Range, with few breaks, extends from Lake Gogebic in Michigan to Mineral Lake in Wisconsin. The portion of the formation which has been productive on a commercial scale extends from Iron Belt, Wisconsin, to the Castile mine at Wakefield. The main footwall dips almost uniformly to the north at an angle of from fifty to seventy degrees. The basal rock is an Archean granite and schist; overlying this, above a thin conglomerate series, is the quartzite foot of the iron formation; above this the lower jaspers, which form the lower ore-bearing portion of the formation. Above the lower jaspers is a slate formation which is again overlain by a heavy jasper formation. Above the upper jaspers is a slate series called the "Tyler" slate, which is most widely developed from Ironwood to the western end of the range. On the eastern end of the range, the overlying slate is missing, having been removed by erosion before the eruption of the Keweenawan, or "Copper Country," trap flows, which overlie the range on the north. The main concentration of orebodies is divided into two classes; the Primary and the Secondary concentration. Primary concentration is most commonly found close to the quartzite or the lower slate, and forms a hard blue ore, which is usually quite narrow.

Secondary concentration, which has formed the largest orebodies on the range, has occurred along the troughs formed by the intersection of dikes with the footwall quartzite or the lower slate. These dikes are the channels through which the Copper Country rock flowed out. The dikes generally pitch to the east and towards the foot.

To the eastward of Black River, the formation is considerably broken up by folds and faults, which has as yet not been thoroughly worked out. In this territory there is one authentic case of a considerable orebody concentrated on a fault trough formed by the intersection of a fault with the footwall. On the east end of the range the lower jasper immediately overlying the quartzite is unproductive, the ore forming on or above the lower slate, which is much thicker than

*General Engineer, Pickands, Mather & Co., Gogebic Range

it is from Ironwood west. The lower jasper is the main ore-bearing zone on the western part of the range.

CHANGES SINCE 1910.

The list of operating properties on the Gogebic Range has been reduced since the last meeting of the Institute here in 1910, by the closing of the Iron Belt mine of The Cleveland-Cliffs Iron Co., in the fall of 1911, and of the Atlantic mine and the Plumer exploration, of the Oliver Iron Mining Co., and the abandoning of the Pence, Hennepin & Snyder exploration, of the Jones & Laughlin Co., west of Montreal. This leaves the Montreal as the operating mine farthest west on the range. There have been several new strikes made on the range in the same period, however.

The largest and most notable of these is the Wakefield mine, which first shipped ore in the summer of 1913. The Wakefield orebody is far south of what was formerly regarded as the main ore-horizon of that part of the Gogebic Range. The Wakefield orebody is one of the largest single deposits so far developed on the range, and is the only one to which open-pit work has been found applicable on a large scale.

A large orebody has been developed in the Palms and Anvil mines of the Newport Mining Co. at a depth of about 1500 feet. The Puritan mine of the Oliver Iron Mining Co. has become a steady producer since 1910, having struck a large orebody. The westerly continuation of the main Newport orebody, mentioned in the program of the 1910 meeting of the Institute, has been developed since that date by the Pabst mine. Explorations near Gogebic Lake and near Mar-enisco, on the east end of the range, and near Mellen on the west, have so far failed to add any new producers.

Several new shafts have been sunk or are in process of sinking on the range. Easternmost of these is the Meteor exploration shaft of Oglebay-Norton & Co., Wakefield, Michigan. This shaft is an incline steel shaft, sunk in the foot-wall to explore the formation east of and below the Castile mine. A vertical steel shaft has been sunk at the Palms mine to develop the new orebody found there. Another vertical footwall shaft is now being sunk at the Newport mine a short distance east of the present main shaft.

The vertical shafts have the disadvantage of increasing length of crosscuts with depth, which is believed to be more

than counterbalanced by the lessened cost of maintenance over an incline shaft and by the greater speed of hoisting which is made possible.

Since 1910, the new "C" shaft of the Norrie mine has been put in service by the Oliver Iron Mining Co. This is an inclined steel shaft in the footwall. It was sunk 840 ft. full size, to meet a cribbed raise 374 ft. high, which was then stripped down. At the Cary mine, of Pickands, Mather & Co., Hurley, Wis., the "A" shaft was put into commission in January of the present year. This is a five-compartment steel incline shaft in the footwall. It was put down by raising and stripping to the 19th level and sunk to the 20th level, which is at a depth of 1290 feet. A similar shaft is partly finished at the Windsor mine, also of Pickands, Mather & Co., which is idle at present. The Ottawa mine of Oglebay-Norton & Co., at Gile, Wisconsin, is raising at several points in the footwall for a new incline shaft.

An interesting development since 1910 is the increasing use of electric power, not only for haulage and lighting but for other purposes about the mines. Some of the mines, like the Wakefield Iron Co., the Newport Mining Co., and the Oliver Iron Mining Co., prefer to manufacture their own power. Several of the other properties buy their power from the Gogebic & Iron Counties Railway & Light Co.

The Castile mine, of Oglebay, Norton & Co., has, besides haulage and lighting equipment underground, an electric pump on the bottom level throwing to a steam pump about seven hundred feet above the bottom. The electric pump is a Prescott plunger pump, rated at 800 gallons capacity per minute for 700 ft. ahead, driven by a 2200-volt alternating-current Allis-Chalmers motor. The Cary mine is about to install an electric pumping plant in its "A" shaft. The Ottawa mine is using an electric compressor. The Wakefield Iron Co. was the first company to use electricity for hoisting ore on the Gogebic Range.

Having given a summary of the conditions on the Gogebic Range in general, we will now take up a description of the particular properties which it is proposed to visit at this time.

"G" SHAFT, OF THE PABST MINE, OLIVER IRON MINING CO.

The first of these is the "G" shaft of the Pabst mine, operated by the Oliver Iron Mining Co. at Ironwood. This

shaft is 1770 ft. deep, on a 64 degree incline. Hoisting is done in two 7-ton skips in balance and one cage. The skip-hoist is an Allis-Chalmers, Corliss duplex first-motion, 28- by 60-in., hoist with drum 12 ft. in diameter and 6 ft. of face. The cage-hoist is a Wellman-Seaver-Morgan, Corliss single-cylinder, 24- by 28-in., second-motion hoist. Air is compressed by a Nordberg Corliss cross-compound condensing compressor. Compression takes place in two stages. The steam cylinders are 26 and 52 in. in diameter; the air cylinders are 28 and 45 inches. The stroke is 48 in., and the capacity 5280 cu. ft. per minute. The boiler plant consists of 8 horizontal tubular boilers, 72 in. in diameter by 18 ft. in length.

The shaft house is a thoroughly modern steel structure, a noticeable feature of which is the use of lattice columns and girders instead of built-up channels and I-beams.

NEWPORT MINE, NEWPORT MINING CO.

Hoisting at the Newport mine is done through two shafts, "D" and "K." "D" is the principal shaft, where the main plant is located. A new vertical shaft, called the Woodbury, is being sunk a short distance east of the "D" shaft.

The boiler plant at the "D" shaft consists of five 250-h.p. and one 400-h.p. Wickes vertical boilers, fired by Roney stokers fed from overhead bunkers. At the "K" shaft there are two 250-h.p. and two 100-h.p. Wickes vertical boilers, hand fired. At the "D" shaft power-house the coal is handled by an elevating and conveying system, and the ashes are removed in a car operated by an endless-rope haulage.

The hoisting plant at the "D" shaft consists of a Nordberg simple twin 34 in. and 34- by 72-in. and a Thompson-Greer simple twin 24 in. and 24- by 48-in. The Nordberg hoist has two drums 12 ft. in diameter by 66 in. of face, placed side by side. One drum is keyed to the shaft, the other operated by a clutch. The brake, throttle, reverse, and clutch are all steam operated, and an automatic overwinding device is provided. The normal capacity of this hoist is 350 tons per hour from a depth of 2000 feet.

The Thompson-Greer hoist has two drums in tandem, 8 ft. diameter by 12 ft. face. Both drums are operated by clutches. This hoist is also provided with an automatic overwinding device.

The 22- by 22- by 48-in. Allis-Chalmers hoist, which is to be used as the cage hoist in the Woodbury shaft, is now

used in the sinking. This hoist has two drums 6 ft. in diameter by 5 ft. face, one keyed to the shaft, the other operated by a clutch, and is equipped with an automatic overwinding device. Hoisting at the "K" shaft is done by a Thompson-Greer hoist which is a duplicate of the one at the "D" shaft.

Air is compressed by a two-stage cross-compound Nordberg compressor 18- and 32- by 42-in. stroke steam and 17½- and 29- by 42-in. stroke air, 75 revolutions per minute, with a capacity of 2500 cu. ft. of free air per minute to 90 pounds pressure.

Power for tramming and pumping underground and for lighting, shop and miscellaneous uses on surface, is generated by two reciprocating engine units of 250-kw. and 150-kw. capacity and one mixed-pressure turbine unit of 500 kw. capacity.

The 500 kw. unit is a General Electric, Curtiss turbine, of the mixed-pressure type, direct connected to a General Electric compound interpole generator. This is operated normally at 1500 r.p.m. by exhaust steam from the hoists and the air compressor, through a steam regenerator which stores energy to allow for the intermittent operation of the hoists. This regenerator will furnish low-pressure steam for operation of the generator at full load for three minutes after the hoists are shut down. Any shortage in the supply of low pressure is automatically compensated for by the admission of high-pressure steam through the high-pressure valves. A Wheeler admiralty-type surface condenser with a capacity of 20,000 pounds of steam per hour and a vacuum of 27 inches condenses all steam from the power units. Circulating water is cooled in a Wheeler Banard forced draft cooling tower.

Lubrication is done by a gravity oil system piped to a Turner oil filter, the product of which is used over again. The entire boiler and power plant is equipped with a modern system of indicating and recording meters.

ANVIL-PALMS MINES, NEWPORT MINING CO.

The Anvil-Palms plant was the first plant on the range to compensate for heat losses in long steam lines by superheating, and the first on the range to operate compound condensing hoists.

The boiler plant consists of 3 Edgemoor 450-h.p. three-pass boilers, equipped with Foster superheaters which superheat the steam 100 degrees Fahrenheit. Firing is done by

Roney stokers fed from overhead bins. Steam is generated at 150-lb. pressure. Coal is stocked in a coal dock and drawn through a tunnel running the whole length of the dock. By slides in the roof of the tunnel the coal is fed to a conveyor which carries it to a single-roll crusher, from which a belt conveyor carries it to the coal bunkers. In the belt conveyor is installed a Merrick weightometer which records the total weight of coal hoisted to the bunkers.

The Anvil powerhouse contains a Laidlaw-Dunn-Gordon cross-compound two-stage compressor of 17-in. and 36-in. by 42-in. stroke steam, and 19-in. and 36-in. by 42-in. stroke air, with a capacity of 3000 cu. ft. per minute to 90 pounds per square inch, with a Westinghouse Leblanc jet condenser, working at 27 in. vacuum. The generator is a cross-compound Allis-Chalmers engine, 14 in. and 28 in. by 36-in., direct connected to a 300 kw. 250-volt direct-current Allis-Chalmers generator with a turbine-driven Westinghouse Leblanc jet condenser.

The hoist at the Anvil power house is a twin tandem-compound reversing Nordberg, 20 in. and 37-in. by 66-in., with two 10-ft. drums with 66-in. face. The clutch, reverse, throttle and brake are all operated by oil, pressure being supplied by a small triplex pump and accumulator. The hoist is provided with a safety overwinding device. Condensing is done by a counter-current jet condenser and air pump, water being cooled in a natural-draft cooling tower.

A gravity oil system with filter lubricates all the units in the power house.

The Palms power house is operated by steam piped from the Anvil boiler plant through an 8-in. pipe a distance of 1500 feet. The steam line is supported on steel bents, and insulated by a 3-in. covering of felt and magnesia protected from the weather by a galvanized-iron covering.

The ore-hoist at the Palms is a duplicate of the Anvil hoist described above.

The cage-hoist is a simple duplex 20x20x48-in. Nordberg first-motion hoist, with two 6-ft. drums with 56-in. faces, grooved for 2700 feet of $1\frac{1}{4}$ -in. rope in three and one-half layers, both drums being keyed to the shaft. The governor is two-speed, one speed of 800 ft. per minute being used for handling men, and the other speed of 1500 ft. per minute being used for handling material. The change of speed is effected by change gears, thus making it possible to take ad-

vantage of the cut-off at both speeds. The hoist will handle a live load of 7000 pounds when operating in balance.

The mine water which is used for boiler feed is treated in a Bartlett-Graver water purifier and filter with a capacity of 7500 gallons per hour.

Like the Newport plant, the Palms-Anvil boiler house and power houses are equipped with a modern system of indicating and recording meters.

Each boiler is equipped with a recording thermograph for reading flue gas temperatures, and also with a differential draft gauge. The air compressor, generator and hoisting engines are provided with recording and indicating vacuum and steam gauges. A graphic record is kept of feed-water temperature and also of the amount of water fed to the boilers. A thermograph in the steam line at the Anvil power house and another in the steam line at the Palms power house record the steam temperature at both points. By comparison the steam-line loss may be figured.

A power house log sheet is kept in which hourly readings of steam pressure, vacuum, temperatures of inlet and discharge water from the various condensers, revolutions of the air compressor, load on the power unit, etc., are taken. This together with the recording meters gives a detailed and complete record of the daily operation of the power house. A boiler house log sheet on the same principle gives a record of boiler, stoker, draft and feed-pump performance.

WAKEFIELD IRON CO.

The Wakefield orebody is located on the main quartzite foot, south and east of the Mikado and Asteroid mines, and south of the Village of Wakefield. The concentration is on a heavy dike about perpendicular to the footwall and pitching east, and differs from a typical Gogebic Range deposit only in its considerable width and in being very close to the surface.

The west part of the orebody is developed by an open pit about one-half mile long, 150 ft. extreme width at bottom, 300 ft. extreme width at the bottom of stripping, 600 ft. extreme width at the top, and 100 ft. deep at its deepest part. East of the open pit the ore is developed by two shafts, "A" and "B." The "A" shaft is 250 ft. deep and the "B" shaft is 400 ft. deep. These shafts are mainly exploratory at present, most of the ore being produced from the pit. The "A" shaft

has the additional purpose of draining the open pit, which is done by churn drill holes. These shafts are now operated by temporary steam plants, which will be replaced by electrical plants in the coming fall.

The hoists are second motion, of the usual type for shallow hoisting. The pumping plant in the "A" shaft, which will be replaced by electrical pumps, has a capacity of 1200 gallons. The "B" shaft is pumped by 2 No. 9 Cameron pumps.

A new electrical equipment is being installed to replace the steam altogether. The power house contains two 250-h.p. Wickes water-tube boilers, working at 175-pound pressure, provided with Roney stokers. Steam is superheated by Foster superheaters; a Hoppes primary heater heats the feedwater by using the exhaust steam from the generator. The final heating of the feedwater is done by a Green economizer. The plant burns screenings.

Power is generated by a 750-kva. Curtiss turbo-generator, a special machine arranged to carry the peak load of the hoists, running up to 1125 kva. for ten seconds. There is also a small 100-kva. two-stage Curtiss generator, and room is provided for another special 750-kva. generator to be installed later. These generate alternating current at 2300 volts, which is transmitted to the substations at the "A" and "B" shaft hoist-houses.

The ore hoist at each shaft is a double-drum Nordberg, run by a 250-h.p. induction motor. The cage hoist is a single-drum 150-h.p. Nordberg hoist. The compressor is located at the "B" shaft hoist-house. It is rope driven by a 325-kva. synchronous motor, and has a capacity of 1800 cu. ft. per minute. Electric pumps will be installed also at the two shafts. The electric plant is not in operation yet.

THE CUYUNA IRON ORE DISTRICT, MINNESOTA.

The Cuyuna Iron Ore District, Minnesota, is the infant district of the Lake Superior region. While its existence was predicted as far back as 1885, by that eminent Wisconsin State Geologist, R. D. Irving, and the map published by him in the United States Geological Survey Monograph No. 19, page 534, it was not until the year 1903, eighteen years afterward, that the first drilling was done and iron-bearing formation actually encountered.

The reason for this speculation was the trough-like or

synclinal structure of the Lake Superior basin, ascertained by the study of all the other iron ore districts, and the reason for the delay of discovery was the absence of rock outcrops in the Cuyuna district. The nearest rock exposures seemingly only complicated matters, although now their relationship is fairly well understood.

Drilling, however, was not the first operation that established the district. The tracing of the iron-bearing formation was first accomplished magnetically and then drilling followed, and the magnetic belts today still largely outline the district. One is, therefore, enabled to say that the Cuyuna district occupies parts of Aitkin, Crow Wing and Morrison counties and probably Todd and Cass counties should be included, but the productive part is entirely in Crow Wing county, the geographical center of the State of Minnesota. This gives a length over all exceeding 60 miles, measured in NE-SW direction, and twice that distance when duplications by folding are also counted.

The areas for exploration are numerous but always long and narrow, and the iron-bearing formation is always located under the magnetic belts or on extensions along the strikes of the belts. The orebodies can be located only by drilling.

The first shipments were made in 1911, amounting to 147,431 tons. In 1912 the shipments increased to 305,000 tons, in 1913 to 733,000 tons, in 1914 to 872,000 tons and for 1915 there is every reason to believe that about 1,250,000 tons will be shipped, without having all the mines in fullest operation.

(A full description of the range is given in paper by Carl Zapffe).

IRON ORE SHIPMENTS FROM GOGEBIC RANGE.
(From Iron Trade Review).

Mine.	1914.	All Years.
Anvil		831,361
Ashland	133,250	6,117,680
Asteroid	135,119	268,346
Atlantic		1,888,820
Brotherton	47,662	2,186,869
Cary	68,464	3,459,943
Castile	36,569	309,909
Colby	291,947	3,529,617
Eureka	23,430	706,062
Geneva		31,303
Harmony		470,260
Iron Belt		1,254,937
Ironton	51,138	1,412,665
Keweenaw	5,771	5,771
Mikado	2,094	1,085,005
Montreal	229,559	3,841,732
Newport	707,485	10,415,729
Norrie Group	984,242	30,258,257
Ottawa	106,260	877,568
Palms	174,177	1,586,900
Pence		91,314
Pike		102,056
Plumer		98,631
Puritan	58,410	373,147
Royal	11,686	22,345
Sunday Lake	54,327	1,821,844
Tilden	114,767	5,607,450
Wakefield	313,050	328,311
Winona		10,500
Yale	19,075	821,699
Shipped prior to 1914 (idle mines).....		939,401
Totals	3,568,482	80,845,441

IRON ORE SHIPMENTS FROM CUYUNA RANGE.
(From Iron Trade Review).

Mine.	1914.	All Years.
Armour No. 1		154,626
Armour No. 2	283,565	508,261
Barrows	47,350	56,439
Cuyuna-Mille Lacs	51,292	75,726
Ironton	40,425	43,361
Kennedy	179,885	790,992
Pennington		101,136
Rowe	78,685	78,685
Thompson	178,202	235,741
Totals	859,404	2,044,967

PRODUCING MINES OF THE GOGEBIC RANGE.

Mine.	Location.	Operator.	Manager.	Superintendent.
Wakefield	Wakefield, Mich.	Wakefield Iron Co.	J. D. Ireland*	W. C. Hart
Castile	Wakefield, Mich.	Oglebay, Norton & Co.	E. W. Hopkins.	P. S. Williams
Sunday Lake	Wakefield, Mich.	Pickands, Mather & Co.	L. M. Hardenburgh*	W. J. Davies
Brotherton	Wakefield, Mich.	Pickands, Mather & Co.	L. M. Hardenburgh.	W. J. Davies
Eureka	Ramsay, Mich.	Oglebay, Norton & Co.	E. W. Hopkins.	P. S. Williams
Anvil	Bessemer, Mich.	Newport Mining Co.	E. L. Cullen.	
Keweenaw	Bessemer, Mich.	Newport Mining Co.	E. L. Cullen.	
Palms	Bessemer, Mich.	Dunn Iron Mining Co.	E. L. Cullen.	
Tilden	Bessemer, Mich.	Oliver Iron Mining Co.	O. C. Davidson*	D. E. Sutherland
Colby	Bessemer, Mich.	Corrigan, McKinney & Co.	W. J. Richards.	G. S. Barber
Yale	Bessemer, Mich.	Chas'l Iron Co. of America.	Geo. J. Webster.	W. E. McRandle
Newport	Ironwood, Mich.	Newport Mining Co.	E. L. Cullen.	
Pabst	Ironwood, Mich.	Oliver Iron Mining Co.	O. C. Davidson*	D. E. Sutherland
Aurora	Ironwood, Mich.	Oliver Iron Mining Co.	O. C. Davidson*	D. E. Sutherland
East Norrie	Ironwood, Mich.	Oliver Iron Mining Co.	O. C. Davidson*	D. E. Sutherland
Norrie	Ironwood, Mich.	Oliver Iron Mining Co.	O. C. Davidson*	D. E. Sutherland
Ashland	Ironwood, Mich.	Hayes Mining Co.		Robert King
Cary	Hurley, Wis.	Pickands, Mather & Co.	L. M. Hardenburgh*	J. M. Davis
Ottawa	Gile, Wis.	Oglebay, Norton & Co.	E. W. Hopkins.	F. B. Goodman
Montreal	Montreal Wis.	Oglebay, Norton & Co.	E. W. Hopkins.	F. B. Goodman

IDLE MINES OF THE GOGEBIC RANGE.

Mine.	Location.	Operator.	Manager.	Superintendent.
Meteor	Wakefield, Mich.	Oglebay, Norton & Co.	E. W. Hopkins.	P. S. Williams
Pike	Wakefield, Mich.			
Chicago	Wakefield, Mich.	Oliver Iron Mining Co.	O. C. Davidson*	D. E. Sutherland
Pilgrim	Wakefield, Mich.	Pickands, Mather & Co.	L. M. Hardenburgh	
Mikado	Wakefield, Mich.	Pickands, Mather & Co.	L. M. Hardenburgh*	
Asteroid	Ramsay, Mich.	Oglebay, Norton & Co.	E. W. Hopkins.	P. S. Williams
Jackpot	Bessemer, Mich.			
Winona	Bessemer, Mich.	Corrigan, McKinney & Co.	W. J. Richards.	G. S. Barber
Ironton	Bessemer, Mich.	Corrigan, McKinney & Co.	W. J. Richards.	G. S. Barber
Bonnie	Ironwood, Mich.	Newport Mining Co.	E. L. Cullen	
Germania	Hurley, Wis.	Harmony Mining Co.		Robert King
Windor	Hurley, Wis.	Pickands, Mather & Co.	L. M. Hardenburgh*	J. M. Davis

MINES BEING DEVELOPED.

Puritan	Bessemer, Mich.	Oliver Iron Mining Co.	O. C. Davidson*	D. E. Sutherland
Geneva	Bessemer, Mich.	Oliver Iron Mining Co.	O. C. Davidson*	D. E. Sutherland
Davis	Bessemer, Mich.	Oliver Iron Mining Co.	O. C. Davidson*	D. E. Sutherland
Berkshire	Mellen, Wis.	Berkshire Mining Co.	Chas. Schultz†	Verne Prisk
*General Superintendent.		†President of Company.		
			‡M. A. Hanna & Co..	Sales Agents.

**LAKE SUPERIOR IRON ORE SHIPMENTS FROM THE DIFFERENT
RANGES FOR YEARS PRIOR TO 1914, 1914, AND GRAND
TOTAL FROM 1855 TO 1914, INCLUSIVE.**

(Compiled from Report Published by Iron Trade Review).

	Prior to 1914.	1914.	Grand Tot.
Marquette Range .. (Tons.....	107,298,812	2,491,857	109,709,669
(Per cent..	17.1	7.6	16.6
Menominee Range ..(Tons.....	89,039,011	3,221,258	92,260,269
(Per cent..	14.3	9.8	14.1
Vermillion Range ..(Tons.....	34,829,073	1,016,993	35,846,066
(Per cent..	5.6	3.1	5.5
Gogebic Range(Tons.....	77,276,959	3,568,482	80,845,441
(Per cent..	12.4	10.9	12.3
Mesabi Range(Tons.....	313,105,968	21,465,967	334,571,935
(Per cent..	50.2	65.7	51.
Cuyuna Range(Tons.....	1,185,563	859,404	2,044,967
(Per cent..	.2	2.6	.3
Miscellaneous (Tons.....	1,336,979	105,756	1,442,744
(Per cent..	.2	.3	.2
Total tons	624,072,365	32,729,726	656,802,091
Total shipments 1913.....		49,947,116	
Decrease in 1914		34.4	

PAPERS

SINKING OF THE WOODBURY SHAFT AT THE NEWPORT MINE, IRONWOOD, MICHIGAN.

BY J. M. BROAN, IRONWOOD, MICH.*

The Woodbury Shaft, which is now being sunk by The Newport Mining Company at Ironwood, Michigan, is located 100 ft. to the foot of the contact at surface, between the iron bearing formation and the underlying sedimentaries. The shaft is vertical, with its length at right angles to the strike of the formation. The overall dimensions are 13 ft. 1 in. by 21 ft. 1 in., having six compartments, which will accommodate 2 skips, 2 cages, 1 ladder road, and necessary piping.

The first 100 ft. of sinking was in quartzite; from 100 ft. to 715 ft. were alternate strata of gray and red slates and quartzite. Below this is granite.

Surface Equipment—The surface equipment, as much as possible was complete before sinking operations were started; that is, the headframe, trestles, hoists and compressors were all in readiness. The headframe, which is temporary, is 60 ft. high and built of timber, with the trestle so arranged as to accommodate both north and south ends of the shaft, thereby making it possible to hoist rock in any of the hoisting compartments and dispose of it by means of a haulage motor and car to a common stockpile. A glance at Fig. 1 will readily explain the arrangement mentioned.

During the sinking of the first 700 ft., practically all of the hoisting was done in 3 compartments by means of 1 single drum hoist, operated by a 50-h.p. motor, and 1 double drum hoist, operated by a 70-h.p. motor. While sinking this portion of the shaft, a duplex horizontal steam hoist was being installed by the Allis-Chalmers Company for handling the cages in the permanent lay-out, and about July 1st, 1915, was put into commission to handle a bucket in the fourth compartment. This hoist being built for a greater load than

* Mining Engineer, Newport Mining Co.



WOODBURY SHAFT HEADFRAME AND SHAFT CREW.



FIGURE 1. GENERAL VIEW OF THE WOODBURY SHAFT HEADFRAME AND TRESTLE ARRANGEMENT.

the temporary hoist, was fitted with a bucket of greater capacity.

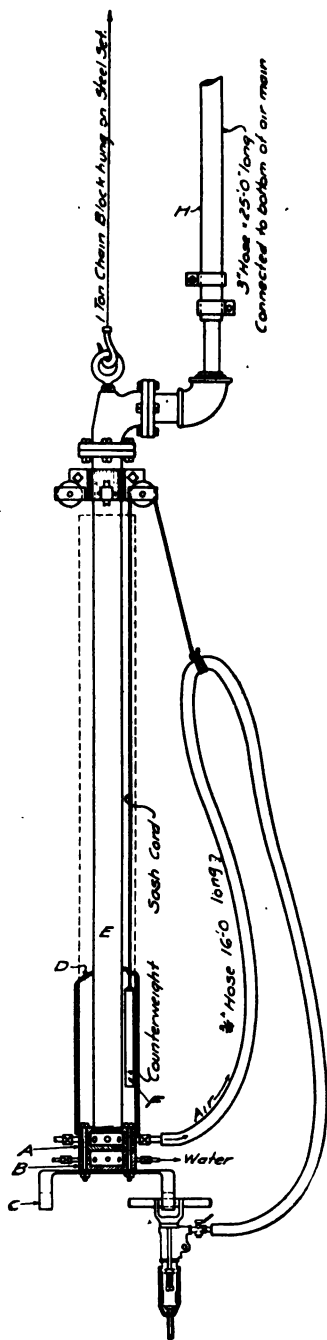
Compressed Air—Three belt-driven compressors, of the Ingersoll-Rand Imperial type No. 10, each driven by a 50-h.p. motor, furnish air at 90- to 100-lb. pressure to 12 jackhammer machines. The air line, which is a 6-in. wrought-iron pipe, is for permanent use.

Drilling Equipment—The first step in the sinking operation is drilling. The equipment for this is, in some respects, different from that used in ordinary practice. First of all the "Header," shown in Fig. 2, by photo and by sketch, distributes air to the machines. While the photo shows the assembled apparatus, the sketch will probably show more distinctly the manner in which the air reaches the machines. A single machine is here shown hanging in position, out of the way when not in use. When ready to drill, all that is necessary is to remove the jackhammer from the hook and pull downward, the counter weight "F" keeping the slack hose out of the way while drilling. In Fig. 2-B "A" is a casting 9 in. in diameter, bored out in the center, and having a bolt circle of a standard 4-in. flange.

Eight holes evenly spaced are drilled in the sides and tapped for $\frac{3}{4}$ -in. nipples, to which the machine hose connections are made. "B" is a duplicate of "A" with the exception that the holes for the nipples are of different size. There are 7 $\frac{1}{2}$ -in. connections and 1 1-in., the latter being an inlet for the water and the others for water discharges to the drills. To be used only with water tube type jackhammers. The hooks or hangers marked "C," are made of $\frac{3}{8}$ -in. by 2-in strap iron. There are four straps with a hook on each end. "A," "B," and "C" are all held together by 4 $\frac{3}{4}$ -in. bolts passing through the 4 in. flange at the bottom of the 4 in. air pipe "E." The ell at the top is made special, with a lug cast on it to accommodate the 1-in. eye bolt by means of which the "Header" is suspended. "D" is a 9-in. pipe which serves as a casing to enclose the counter-weights "F." Two of these headers are used. Each will accommodate 7 jackhammers and 1 blow-pipe, but only 6 machines are used on each at present. When not in use the "Headers" are hung off to one side in the headframe and can be easily lowered by means of a sling beneath the bucket. While in use they hang on a small chain-block fastened to the bottom shaft set. By means of this chain-block the apparatus can be brought to any desired height,



FIGURE 2A. ASSEMBLED HEADER.



GENERAL ARRANGEMENT OF
DRILLING APPARATUS IN
WOODBURY SHAFT
THE NEWPORT MINING CO.
IRONWOOD MICH.

Drawn by: J. M. B. Date: 7-12-13.

FIGURE 2B. SKETCH OF THE HEADER.

the adjustment being allowed by the use of a 3-in. air hose "H" which connects to the air main, the bottom of which is always far enough up the shaft to avoid any severe blows during blasting. One of these outfits can be taken from its position on surface and placed on the chain-block below ready for drilling in less than five minutes, only one connection being necessary to make. While in the softer slates water tube pistons were used, and air blown through the same in place of



FIGURE 3. DRILL PULLERS.

water. A short piece of $\frac{3}{8}$ -in. rubber hose, shown at "A" in Fig. 3 delivered air from the $\frac{3}{4}$ -in. air hose to the tube connections. Later when drilling in the granite the drilling speed was not as great and it was found that sufficient air could be supplied through the ordinary piston to clear the choppings from the drill. The water-tube pistons were then replaced by the ordinary pistons, and the by-pass hose disposed of, thus giving greater efficiency in air consumption, but no noticeable decrease in drilling speed. The steel, which is $\frac{7}{8}$ -in. hol-

low hexagon, is made up into lengths varying by 1-ft. changes from 12 in. to 10 ft., with a $\frac{1}{8}$ -in. difference in gauge for each drill; the first bit being $2\frac{1}{4}$ -in.

Until the granite was reached, the four-point or cross-bit was used but the wear on the gauge became such as to warrant a change if something more serviceable could be found. It was then that the Carr bit was tried out. Considerable difficulty was encountered, especially in the stratified rocks with fissured holes and stuck drills. Rather than abandon a hole, much time was often spent in freeing a drill. It was here that necessity lead to the conception of the 2 pullers shown in Fig. 3. In case only a short portion of the steel emerged, the long gooseneck, shown on the left, was used, while if 2 or more ft. of the drill remained out of the hole, the shorter puller shown on the right could be applied. In either case the inverted jackhammer supplied the necessary power to extract the drill. While in the soft slates, 10-ft. drills were used with very little difficulty and sinks measuring as deep as 9 ft. have been blasted successfully. About 470 lin. ft. of drilling was required per cut in these slates and this could be completed in from 4 to 5 hours. In the hard quartzite, dike, and granite the gauge on the steel would not hold up long enough to permit the use of any drill over 8-ft. long.

In these rocks about 425 lin. ft. of drilling is necessary, which can be drilled in from 7 to 8 hours. The breaking of the holes is dependent entirely upon their arrangement and order in which they are fired. Fig. 4 shows the plan and section of the arrangement and order of firing used while in soft slates. Line "AA" represents a bedding plane on which the rows No. 1, 2 and 3 on the right were bottomed. The shaft was blasted in two separate blasts, the first being made on the right, and the holes blasted in the order numbered in the plan. Three holes marked No. 1 were fired simultaneously with No. 6 electric blasting caps. The exploders in the remaining holes were made up of No. 8 caps and electric delay fuse igniters.

A different arrangement of drilling and order of firing has been found more satisfactory in the granite. Fig. 5 shows 2 rows of holes marked No. 1 which are drilled about 5 ft. deep at an angle of 60° ; then 2 rows marked No. 2 about 8 ft. deep at an angle of 70° . These 4 rows of 5 holes each comprise the cutting holes, which are fired in the order num-

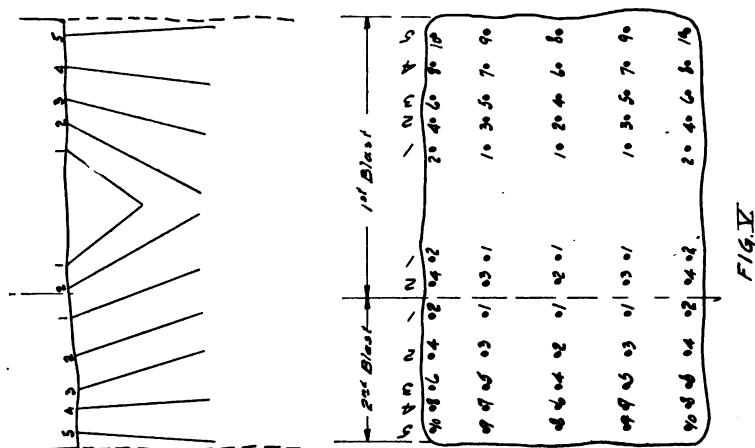


FIGURE 5. ARRANGEMENT AND ORDER OF FIRING OF HOLES IN GRANITE.

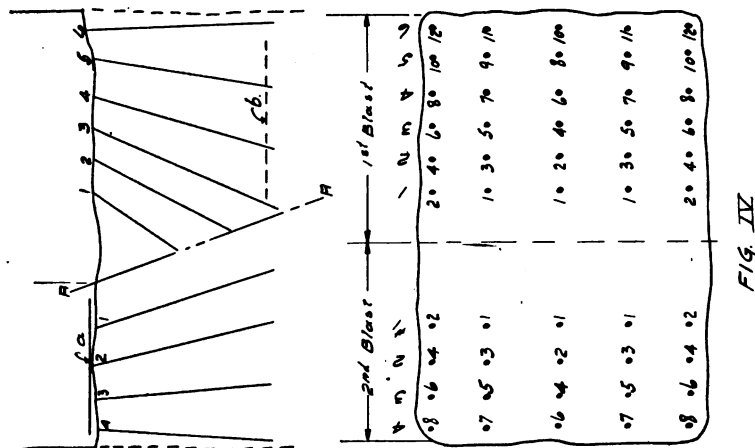


FIGURE 4. ARRANGEMENT AND ORIER OF FIRING OF HOLES IN SOFT SLATE.

bered in the plan. The 6 No. 1 holes are fired with No. 6 electric blasting caps and the others with No. 8 caps and electric delay fuse igniters.

Blasting—Mention has already been made of the order of firing in the different cuts, but before leaving the subject it might be of interest to know a few of the details of the blasting operation. When the shaft was started, DuPont Crescent, Double Tape and Triple Tape fuse were all tried out and it was found that while the Double and Triple tape fuse were more impervious to water, they were too brittle when exposed to cold air and water, and unless handled with great care they would break, thus causing a discontinuation of the powder train. Crescent fuse, however, was found to be sufficiently waterproof for use here as it is very seldom that a fuse is exposed to water more than 15 minutes before ignition. Furthermore, Crescent fuse is much more pliable and can stand more abuse than the others without damaging the powder train. When using fuse of this kind all exploders were made with fuse of the same length and when the holes were all charged they were cut so as to fire the holes in the order desired. The ends of these fuse were then placed in a paste-board box containing a small amount of black powder. An electric fusee ignited this powder which in turn lighted all the fuse simultaneously. As a matter of convenience and saving of fuse, several boxes were used, thus using much shorter fuse than were necessary if a single box was placed in the center of the shaft. The foregoing method seemed to give good results when the ends of the fuse were all kept perfectly dry and other conditions satisfactory, but precaution had to be taken in order to make a success of every blast. Besides this, the smoke problem had considerable to do with the bringing about of a change. In blasting every cut, about 450 ft. of fuse and $\frac{1}{2}$ -lb. of black powder was burned. This gave off more fumes than could be conveniently disposed of as a greater depth was attained and it was then that the DuPont electric blasting caps and electric delay fuse igniters were introduced. These required much less care in handling and gave satisfactory results. In Fig. 6 a DuPont electric delay fuse igniter is shown at the top. This consists of a short piece of fuse, one end of which is inserted in a brass casing containing the ends of two wires connected by a fusible bridge. The passing of a current of about one ampere fuses this bridge which in turn ignites the fuse.

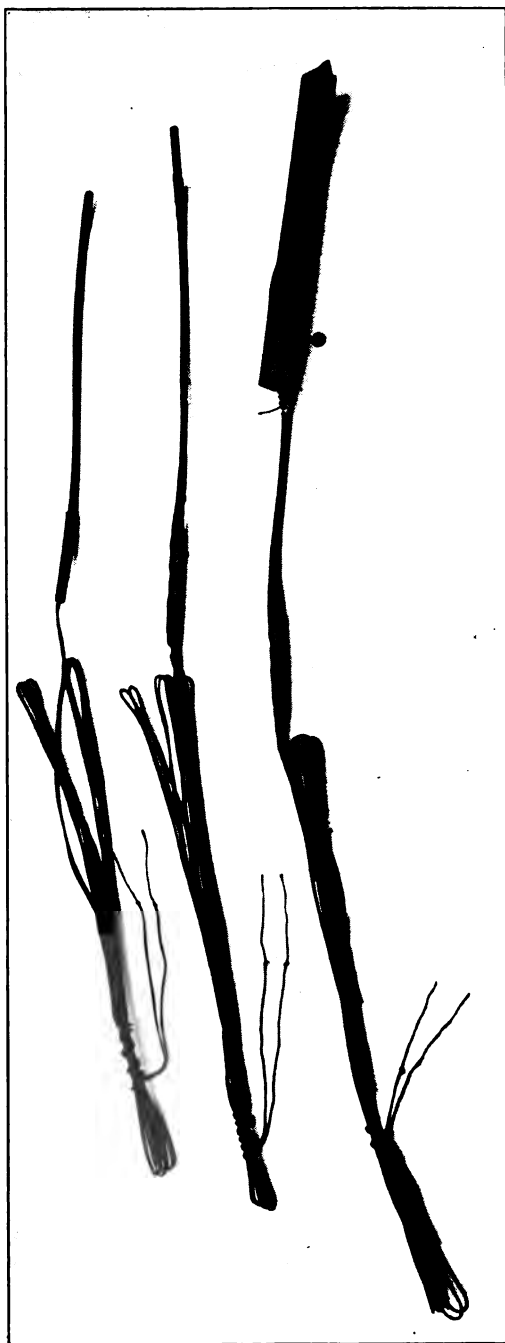


FIGURE 6. DU PONT ELECTRIC DELAY FUSE IGNITERS.

To regulate the time of exploding of a detonator, all that is necessary is to make the fuse of different lengths. The maximum length in which they are manufactured is 12 in., $\frac{3}{4}$ of an in. of this is inside the casing, leaving $11\frac{1}{4}$ in. to be divided into the different delays desired. In several preliminary tests it was found that a $\frac{3}{4}$ -in. delay was a minimum that would give positive results; if cut shorter than this the holes are apt to fire out of order on account of any inaccuracy in cutting and also on account of the 10 per cent. variation in the burning speed of all fuse. On the other end of the fuse in the above described igniter, a No. 8 cap is crimped as shown in the center of Fig. 6. The joints where the fuse enter the brass casing on one end and the cap on the other, are both bound with friction tape, the former then being dipped in melted roofing cement and the latter thoroughly greased, to insure a positive resistance to water. This method has been accepted as the best, after considerable time was spent in experimenting along this line. With this much completed the detonator is then placed in a cartridge of 60 per cent. nitro-glycerine dynamite and the shell tied with a string securing the joint as shown at the bottom of Fig. 6. This joint is also well greased.

Since the steel used is only $\frac{7}{8}$ -in. in diameter, the bottom of a deep hole is, of course, very small and can not contain sufficient explosives to break the burden in a satisfactory manner. To offset this disadvantage as much as possible, without drilling more holes, two or three sticks of 100 per cent. blasting gelatine is placed in the bottom of each hole. The remainder of the charge with the exception of the cartridge containing the detonator is 80 per cent. blasting gelatine. Experiments by the DuPont people have shown that 60 per cent. nitro-glycerine gives a maximum efficiency for speeding up the action of a charge, and for this reason a single cartridge of this strength is used to contain the detonator. It is placed as near the top of the charge as considered safe from being cut off by the breaking of an adjacent hole. Cartridges of sand are used for tamping. An average of 25.50 lbs. of powder per foot of shaft, or approximately 2 lbs. per cubic yard of solid rock, has been used in the first 1100 ft. of shaft.

In preparing a blast, the leads, which are No. 20 copper wire, are laid over the center of the portion to be blasted and the various igniters connected in parallel to them. The main reason for using this method of connection being to prevent

a misfire in case of a single defective igniter or connection, which would cause the failure of the blast if connected in series. It is safe to figure about one ampere per path. From 30 to 35 holes is about the maximum number fired at one time. The switch for closing the circuit is locked in a small cupboard on surface, and the key is kept by the shift boss during the preparation of the blast. Every igniter is tested with a small galvanometer before being used.

Ventilation—After a blast the smoke is cleared away by means of a draft forced through a 12-in. pipe by a 7-h.p. fan, on surface. At first the fan was used to draw the smoke out, but by reversing the air current a marked advantage was noticeable in the time required to clear the smoke.

Mucking—During the first 700 ft. of sinking the rock was disposed of by means of three buckets having a capacity of 26 cu. ft. each. In order to make shoveling as easy as possible, steel plates have been used as sollars, with considerable success. In Fig. 4 "A" shows the position of these plates, when the first blast is made. The breaking of the holes in this blast has a tendency to throw the rock to the position of the plates. In the second blast of the same cut the plates are placed in position "B" Fig. 4.

Shaft Construction—During the mining operations below, the construction crew is employed placing sets, ladders, etc., above. The essential features of the shaft construction are shown in Fig. 7 in plan and elevations. The steel sets are made of 6-in. "H" sections (23.8 lbs.) and hung on studdles made of 3-in. by 5-in. angle iron. Sets are spaced 6 ft. centers in the slates, and 8 ft. centers in the granite. At about every 100 ft. of shaft a bearing set as shown in Fig. 7 is placed under the shaft set. These are supported temporarily by large spruce sprags about 12 or 15-in. in diameter; which will be replaced later by concrete. While in the slates the steel sets could be kept within 15 ft. of the bottom without being damaged, but in the granite if hung closer than 30 ft. to the bottom, blasting is liable to do considerable harm. To date very few pieces have been replaced on this account. The staging used to work on is made of 2-in. hardwood plank, supported by pieces of 2½-in. extra heavy pipe. These pipes are hung in the form of slings by means of pieces of ¾-in. steel rope fastened to the ends and hooked over the flange of the "H" section on the shaft set above. There are 5 men on the crew which place these sets. These men can hang a

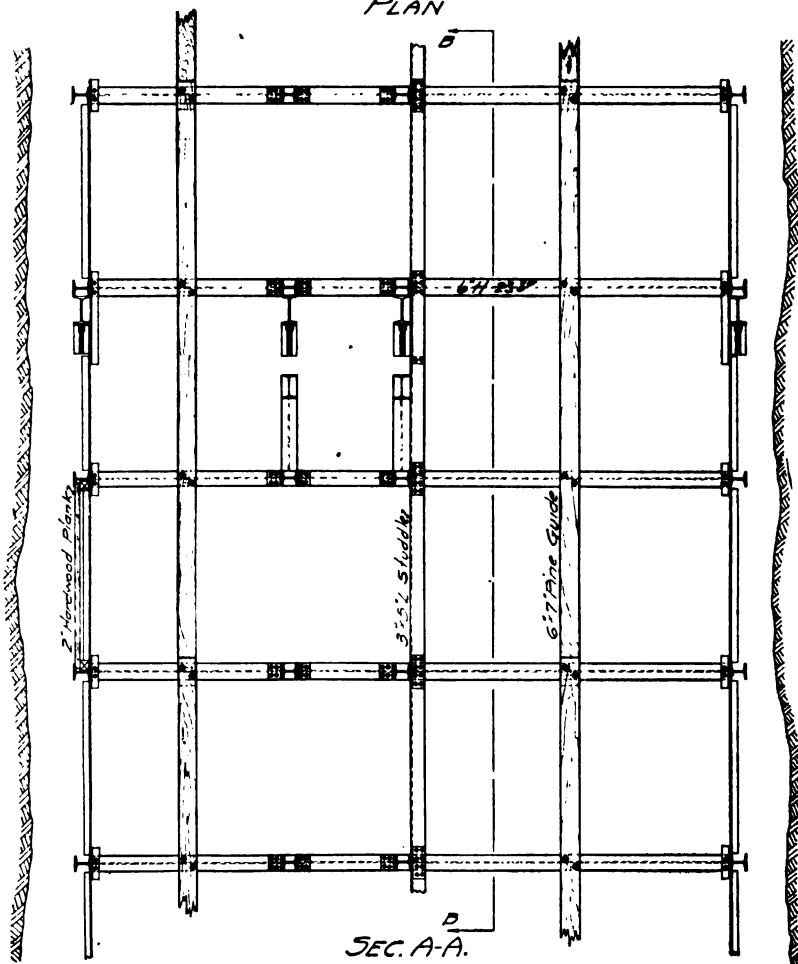
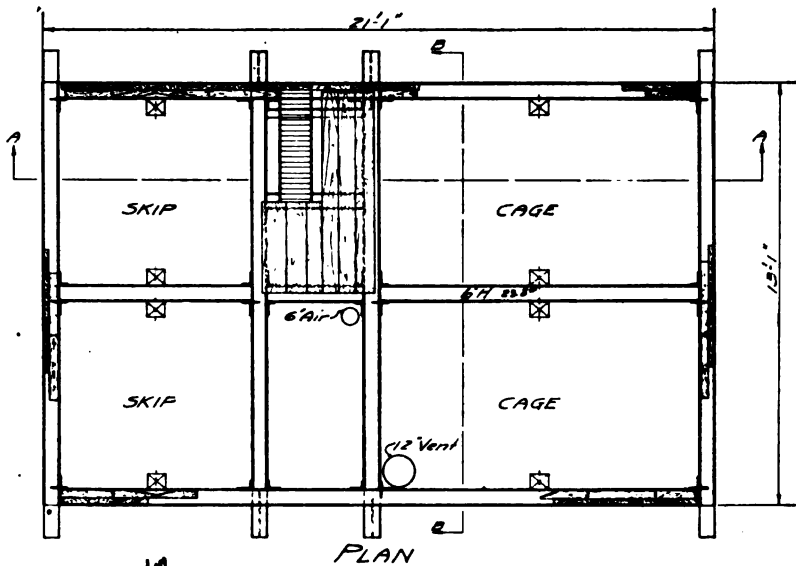


FIGURE 7. WOODBURY SHAFT CONSTRUCTION.
Illustration is one-third size of original drawings.

WOODBURY SHAFT

THE NEWPORT MINING COMPANY

IRONWOOD MICHIGAN

Drawn by: J.M.B.
 Traced by: R.N. Scale: 1/2" = 1 ft
 Checked by: G.M.S. Date: 7-3-15
 Approved: J.M.B.
 Chief Engineer

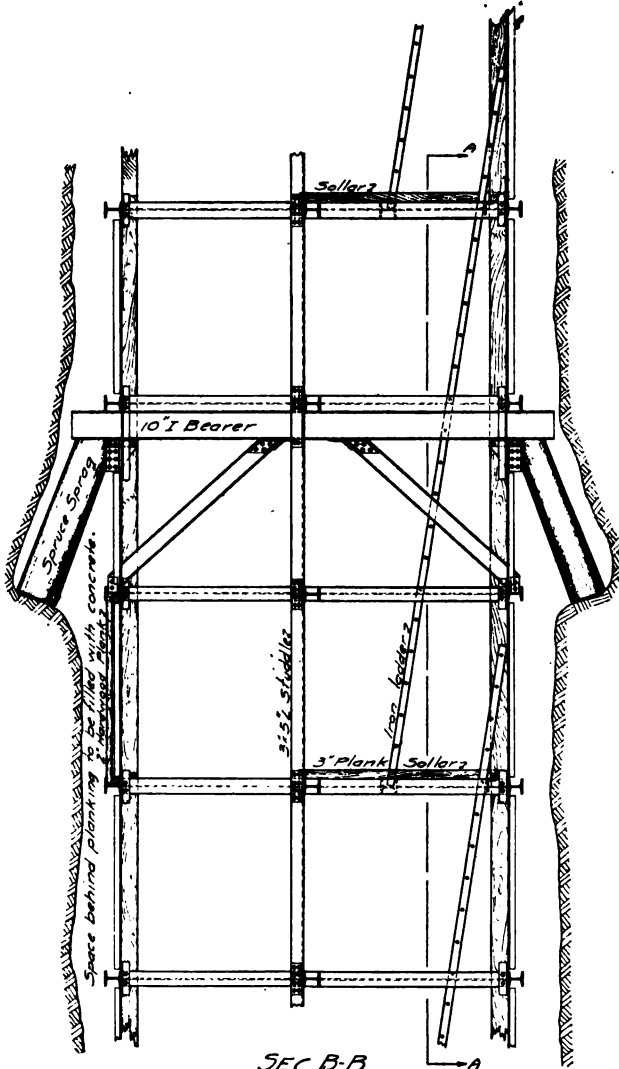


FIGURE 7. WOODBURY SHAFT CONSTRUCTION.
 Illustration is one-third size of original drawings.

set and put in the wooden lath in an 8-hour shift. On the second and third shifts there are construction crews of three men each who place guides, ladders, and sollars, and complete any work the first crew may have left undone.

Guides are made of 6-in. by 7-in. pine, dressed, and are framed to span 2 8-ft. sets or 3 6-ft. sets. Lath for lining the shaft are made of 2-in. hardwood. These will also serve as forms for concreting when sinking operations are completed. The ladders are made of $\frac{3}{8}$ -in. by 2-in. strap iron sides and $\frac{3}{4}$ -in. iron rungs. They are long enough to reach over 2 8-ft. sets or 3 6-ft. sets and in both cases have about 4 ft. projecting above the sollar. The sollars are made of 3-in. plank.

Electric Wiring—All lighting is done by electricity. A single lamp is placed under each ladder sollar to light the ladder to the sollar below. At the bottom of the shaft, two clusters of four lights each are hung, one below the staging to give light to the miners and one above to give light to the construction crew. An electric signal system is used, the wires being run down the east side of the shaft beneath the ladders. By means of a jumper connected to a common return wire, either signal bell can be rung from every sollar.

Labor—The men employed in the shaft are as follows: One shaft captain, 3 shift bosses, 36 miners, 11 construction men, 1 electrician, 4 landers, 2 motormen, 4 hoist engineers, a total of 62 men.

Development—The excavation commenced March 1, 1915, and on September 1, 1915, was down to 1129 ft., which is practically one-half of the total depth, 2260 feet. The maximum monthly development was 201 ft., and the minimum was 173 feet. The average footage per 24 hrs. was 6.20.

Safety—The first step along the line of accident prevention was the use of hard hats. In order that no excuse can be made, every man is furnished with a hat made of felt treated with resin and shellac. These hats are very hard and will resist a severe blow. No person is allowed to enter the shaft without one. Danger signs are placed in conspicuous places warning loafers to keep out. Moveable sollars made of steel plates and operated by levers, are placed over the two compartments most used for loading supplies. When the bucket hangs at the brace the lever is thrown and the plates close in around the bucket making practically a complete cover of the compartment. All buckets when lowered from surface

are stopped just above the point where the construction crew is working and are rung down from there by the men below.

Owing to numerous infections in minor wounds, an antiseptic, consisting of a 2 per cent. solution of lysol, has been placed in the wash room used by the shaft men. To date no accidents of a serious nature have occurred.

DISCUSSION.

MR. KELLY: This paper ought not to be passed without some expression of appreciation of the wonderful work which has been done in sinking the Woodbury shaft. We have heard about the remarkable speed in drifting in the West but I don't know that there is any record of sinking in this country as good as this. It far surpasses anything that we have heard of on Lake Superior; of that I am quite sure, and this ought to be emphasized.

MR. HEARDING: Was the shaft sunk dry or did you have to do pumping?

MR. BROAN: We have had practically no water. What little there is we take out with the rock by shoveling. There are no pumps.

MR. HEARDING: You have no pumps in the shaft?

MR. BROAN: No, sir, no pumps.

MR. BUSH: What is the dip of the footwall at the point where the shaft is being sunk?

MR. BROAN: It is from 68 to 70 degrees.

MR. BUSH: How long will your cross-cuts be at the ultimate depth?

MR. BROAN: Figuring a depth of 2400 ft., the cross-cuts will be in the neighborhood of a quarter of a mile—1300 feet.

MR. BUSH: Notwithstanding that fact, you figure that the difference in the maintenance charges will overcome the increased cost of cross-cutting and tramping to the shaft?

MR. BROAN: Yes, sir; we figure the maintenance of this shaft will be practically nothing as compared with that of the present shaft.

MR. BUSH: This shaft is to take the place of the present incline shaft?

MR. BROAN: Yes, sir, the old shaft will be abandoned as soon as the new shaft is completed.

MR. HARDENBURGH: Are there any further questions that any one would like to ask Mr. Broan? I will say, Mr. Kelly, that we are rather proud that this happened on the Gogebic Range.

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3 6000

5. 8

MINING METHODS ON THE GOGEBIC RANGE.

BY COMMITTEE CONSISTING OF O. E. OLSEN, O. M. SCHAUS AND
FRANK BLACKWELL.

Before entering upon a discussion of the methods used in extracting the iron ores of the Gogebic range, a brief description of their nature and occurrence may serve to make the reasons for the particular methods in use, a little clearer.

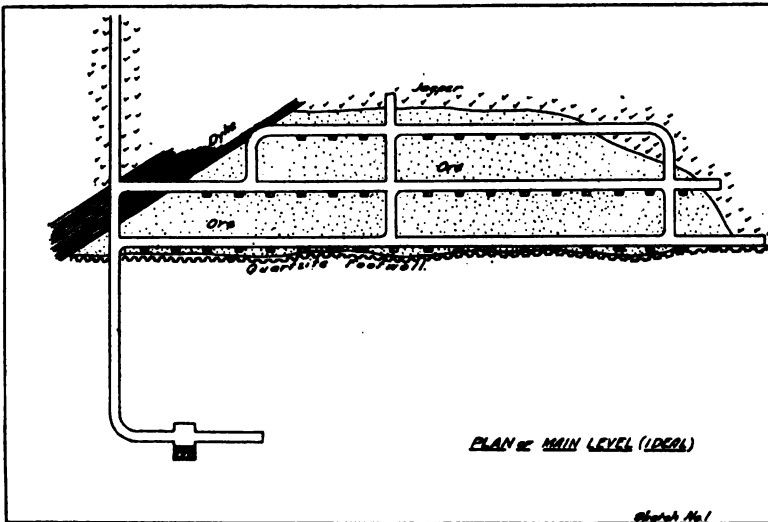
The iron-bearing member is a very regular one, the productive portions of which extend through a distance of about 25 miles east and west. The strike of this formation runs about N. 60 deg. E. at the west end, gradually approaching due east between Bessemer and Wakefield, and tending a little south of east beyond Wakefield. The dip is about 64 deg. to the north, though this will vary locally from 50 to 80 degrees. On the south or footwall side is a fragmental vitreous quartzite, and south of that a quartz slate. The iron-bearing member is made up of orebodies, ferruginous slates and ferruginous cherts, with small amounts of black slate, and some cherty iron carbonate still unaltered. North of the iron formation is a black slate, and beyond this, the trap rock. Cutting through these formations are numerous diorite and diabase dykes, which generally dip at right angles to the footwall, and pitch downward toward the east, though there are several cases where the pitch is toward the west. The footwall quartzite, the ferruginous slates and the dykes are generally impervious to water. The largest orebodies are usually found in the troughs formed where the dykes cut through these two other strata.

From the uniformity with which these formations run, development work generally consists of drifting on the footwall, with occasional crosscuts through the ferruginous slates and raises wherever it is possible to find a trough formed by a dyke and the two other formations. Wherever orebodies occur in these troughs, the ore will usually be a soft ore, and the large bodies of the district are of this nature. In some

mines the bodies occur as narrow veins, lying either on the quartzite or on the slate. These veins usually extend to greater heights than those which bottom on the dykes, and are generally made up of hard ore.

The principal methods of mining are sub-level slicing, which is general in soft orebodies, and various forms of back and underhand stoping, and milling, which are used in hard orebodies. The sub-level slicing method, by which the greater bulk of the ore is mined, will be considered first.

In the determination of the limits of the orebody, a drift is driven on the strike of the formation, a short distance from

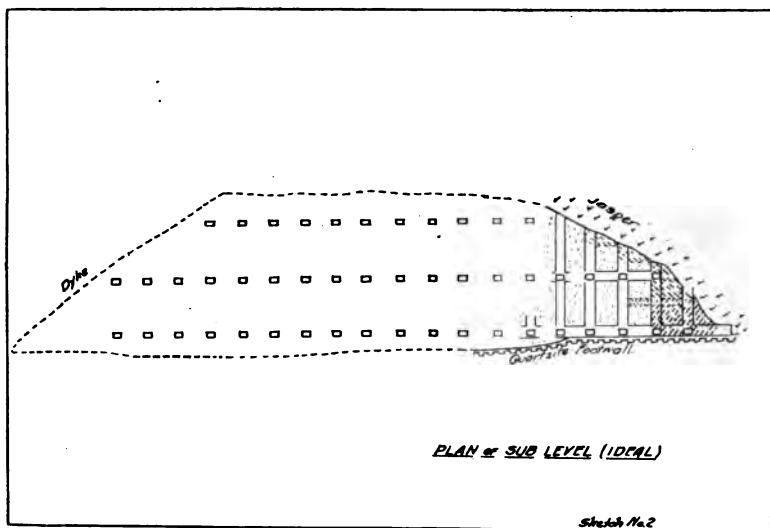


the footwall, and carried to the end of the ore. Crosscuts from 100- to 300-ft. apart are driven to the capping. If the width of the ore permits, one or more drifts are driven parallel to the footwall, leaving pillars of about 50 ft. between. As this drifting progresses, a few raises are put up to test the height of the ore, if it is a new orebody. After the general limits of the ore are determined, a regular series of raises are put up, commencing at the end farthest from the shaft. The raises are spaced from 35 to 50 ft. apart, and are placed in line north and south upon the several drifts. (See sketch No. 1). These raises are 3- by 7-ft. inside of timbers. They are lined with 6-in. cribbing, which is cut to a length of 3 ft.

between joggles, so that all pieces are interchangeable. Double dividers are used between the ladder and chute compartments, so that if the chute side wears out, the repairs are much simpler.

When these raises reach a height of 20- to 23-ft. above the floor of the level, a sub-level set is put in, and the raise is continued upon the back of this set to a height of from 17- to 20-ft. above the floor of the first sub, when another sub-level set is put in. This is carried out till the top of the ore is reached.

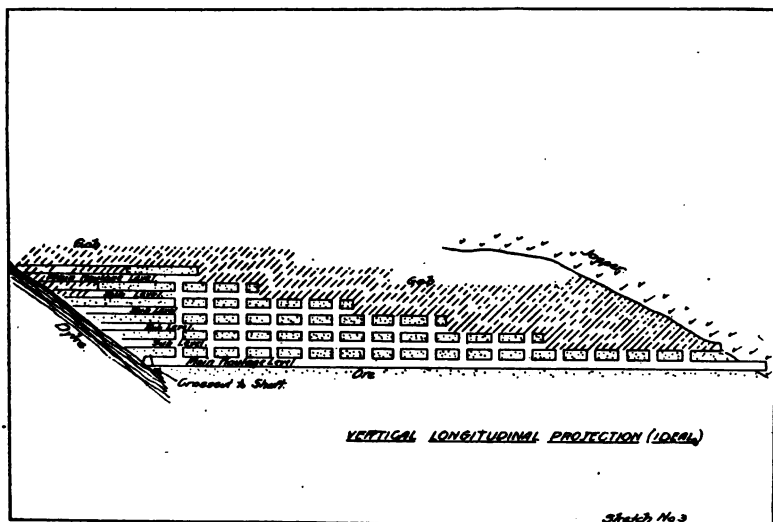
Beginning on the top sub of the line of raises farthest from the shaft, a crosscut is driven from the foot to the



hanging rock. Slicing drifts are driven lengthwise of the orebody, from this crosscut, to the rock that marks the end of the ore. The floor of the slice drift is then covered with boards or slabs, and the back of the drift is caved down, extending up to the capping or to the cave left by the workings above. When one slice drift has been drawn back to the crosscut, another is driven along the side of it, in some instances leaving pillars about 3 ft. between drifts. This pillar is drawn back along with the drift. In some mines the first slice drift is not drawn back till the next drift has been driven along the side of it, when the first drift is caved, leaving the second standing till a third has been driven, and so on. In the mean-

time the next crosscut has been opened, and slicing drifts driven to the cave left by the first crosscut. This block is drawn back in the same manner as the first, and so on back to the shaft.

If the orebody is wide, with several raises in each crosscut to dump into, the slicing would begin midway between raises, and several gangs might work in the same crosscut, each drawing back to its own raise. If the ore is narrow, the caving is begun at the hanging, and carried back toward the foot. (See sketch No. 2). If the raises on the main level are spaced 50-ft. apart, thus making wider blocks on the sub-level, slicing drifts from the first crosscut will be driven to meet those



from the second crosscut, and half the block will be drawn back to each. The next sub below is opened up in the same manner as the top sub, and slicing is begun as soon as the top sub is drawn back to a safe distance. The caving above the second sub is carried up to the boards, with which the bottom of the top sub was covered. These not only permit of a clean extraction of ore, but also indicate that all of the back has been removed. The bottom of the second sub is likewise covered with boards, and the ore is drawn back in the same manner as on the top one. The back of the stope begins to break off and forms what is called the gob, which serves a double purpose, it acts as a cushion against any large falls of ground,

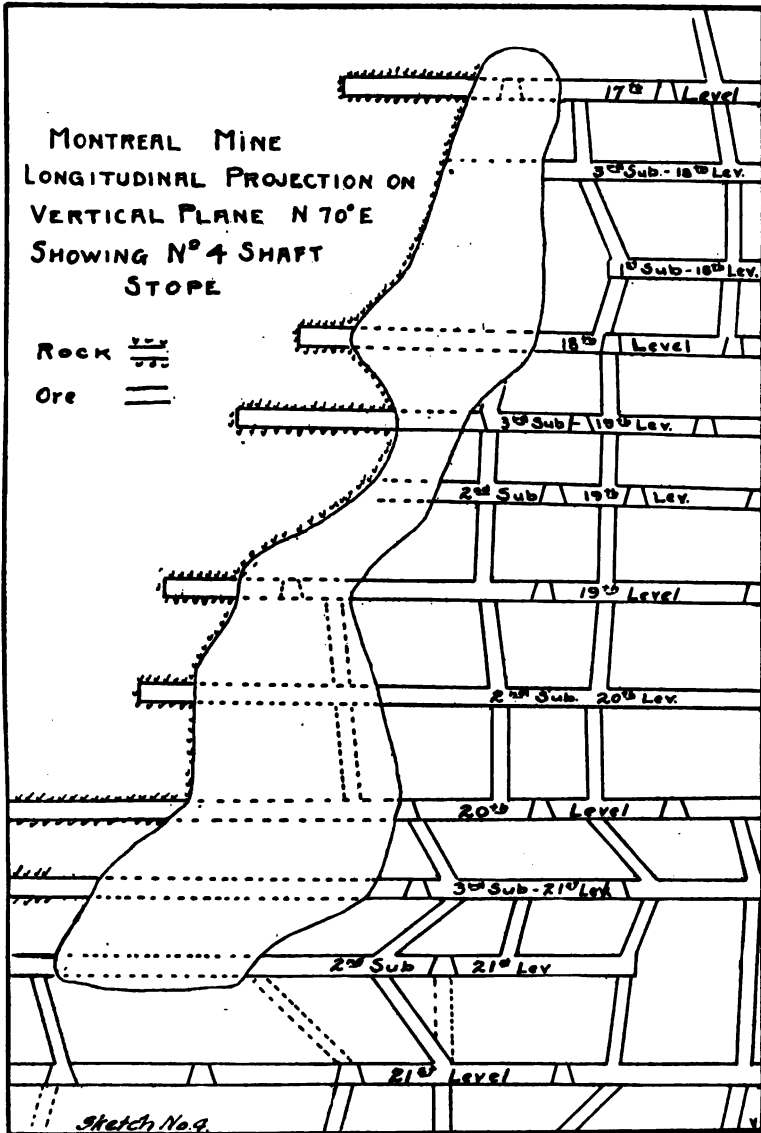
and, on account of the way it holds together, permits the extraction of all ore underneath it, before giving away. The top sub is always carried back a safe distance ahead of the second sub, and the third sub is opened up when the second has retreated a like amount. (See sketch No. 3). This process is repeated until the main level is reached. Instead of trying to draw this, it is allowed to stand until it can be taken from below, when it is treated as a sub-level.

The ore in the subs is trammed in one-ton cars called bug-gies. The chutes are protected by steel rails, laid crosswise and spaced from 8- to 10-in. apart, which guard against anyone falling into them and serve to limit the size of chunks which can be sent to the shaft. The ladder-way is covered with a self-closing door, either latticed or tightly boarded, depending upon the direction of the ventilation.

The item of ventilation has become one of great importance in the mines with large bodies of soft ore which necessitate the use of large amounts of timber. The heat and gases, generated by decaying timber, must be constantly carried away from the working places, in order to enable men to work efficiently.

All the mines have two openings, at least, in one of which the air currents are naturally downcast. By a judicious use of air-tight doors and brattices this fresh air current is deflected to as many working places as possible, and the balance of the openings, that cannot be reached by natural circulation, are supplied with motor driven fans, which render very satisfactory service. The sub-levels are kept connected with the main level above as much as possible, so that timber can be brought from the shaft on the main level and dropped down through the raises to the lower subs, instead of being hoisted up from the main level below. Wherever it is necessary to hoist timber, a small puffer is mounted on a truck which can be transferred to different places, and the timber hoisted by power. In all of the shafts sunk, in recent years, the cage compartment is wide enough, from foot to hanging, to admit of a truck, loaded with timber at surface, being wheeled on the cage and lowered into the mine, thus saving any extra handling.

The chutes of the raises are closed by steel doors, operated by hand levers. The ore is loaded into two-ton saddle back steel cars, hauled by electric motors. At the shaft it is dumped into large storage pockets, beneath which are auxil-



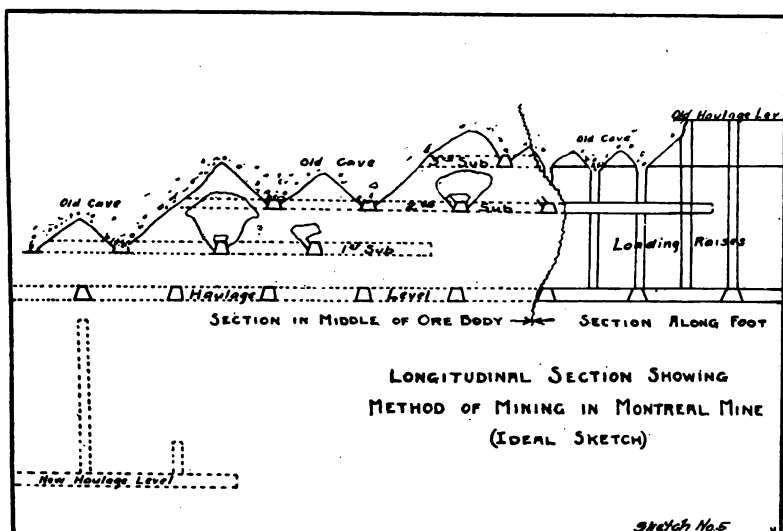
inary pockets that hold one skip load. Skips vary in capacity from 4- to 7-tons. The electric motors are equipped with automatic gongs, which give warning of their approach, when still some distance away. Red lights are carried on the rear cars, and automatic block signals are used at points where two motors are run over the same track. At the chutes, the trolley wire is protected by guards, to prevent the chute tender from coming in contact with it, while filling cars.

In soft ore deposits, all drilling is done with power augers. Three types of bits are commonly used, the diamond point bit for mixed, rubbly ground, the chisel bit for uniform hard ground, and the fish-tail bit for soft ground. The speed of drilling with augers varies from 5-to 14-in. per minute, and a round of hole takes from one to two hours, depending on the ground. Stopping drills are used for back holes and raises. The hollow steel water drill is being used with great success in hard ore and jasper. The heavy reciprocating drills are still used for hard cherty jasper. In shaft sinking in slates and granite the hollow steel jackhammer is used almost entirely. This same type of drill is now being mounted on a carriage, and is a success in putting in horizontal holes.

The sub-level, back-stoping and milling methods are in use at the Montreal mine. The back-stoping is used where the ore is hard and the back of the stope firm, with the orebody not too wide. The main level drift is run along the foot, and crosscuts are driven every 50 ft., with raises on the footwall opposite each crosscut. The main levels are 100 ft. apart, and one to three subs are opened, with drifts along the foot, and crosscuts spaced 25 ft. apart, driven to the hanging. Branch raises, equipped with chute discs, are put up from the main raises, starting about 25 ft. below the sub. Stopping then starts at the hanging, and is carried back toward the foot. Several grades of ore may be stored in the sub-raises, and held until the main raise is clear, thus simplifying the grading.

The sub-level milling system is used where the orebody is wider, and the back not so firm. Better results as to grading are also obtained. Where the orebody is 100 ft. wide or over, foot and hanging drifts are driven with crosscuts every 50 feet. Raises are put up on both drifts, 25 ft. apart. The main levels are 100 ft. apart and three subs are laid out. Beginning on the top sub, from the foot raise nearest the middle of the orebody, a crosscut is driven to the hanging. Raises

are put up on both sides of this crosscut, inclined 45 deg. east and west, and spaced 10 ft. apart. These raises are provided with chutes, but do not need to be cribbed. Similar crosscuts are opened up 50 ft. away on either side, and the raises from these crosscuts meet those put from the first crosscut. The ore is then milled into the chutes and trammed to the main raises. This finally leaves a hog-back between the crosscuts. The crosscuts on the second sub are staggered with respect to those on the top sub, so that the raises from the second sub will come up in the crosscuts of the top sub. The milling from the second sub then takes all of the ore left in the hog-

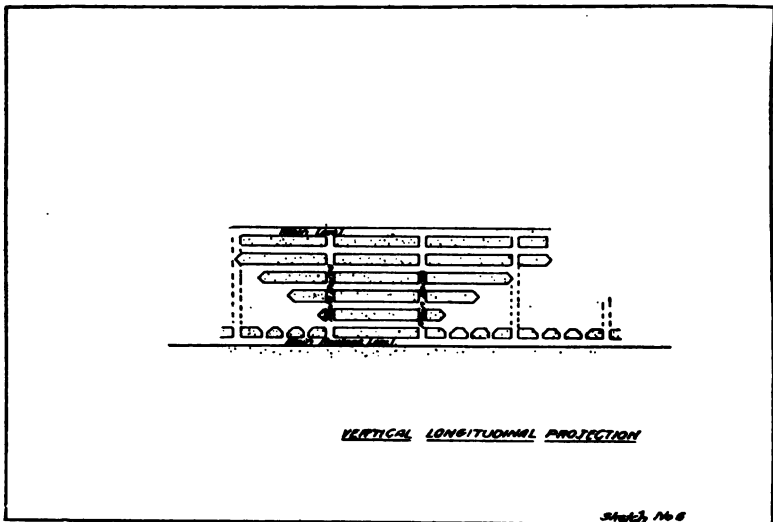


back of the first sub, and leaves a similar one on the second sub. In some places milling starts midway between foot and hanging, and the draw is both ways. The top sub is kept ahead of those below and, when the draw reaches the main raises, the ore is milled directly into them. (See sketch No. 5).

Auger drills are used extensively in the milling system, but the large reciprocating drills are used entirely in back-stoping. The chief difficulty in both of these methods is to keep the main raises in repair, as the wear and tear on the chutes, caused by hard ore, is much greater than in the soft ore mines. There are double dividers between the ladder and

chute compartments and bearing pieces are put in every 25 ft., so that sections of the raise may be repaired without ripping out the entire raise. Where it is necessary to drop the ore any great distance, it is confined in chutes at each level, in order to break the fall and decrease the speed with which it runs through the raises. On account of the small amount of timber used, and the open stopes which are left behind, no trouble is experienced with ventilation.

A combination of sub-slicing and stoping is in use in several mines where the ore is not uniformly hard enough for back-stoping, and where there are irregularities in the width



of the ore. Sub-levels are opened up from a few main raises, leaving a back of ore over each sub of about 10 feet. The sub-drifts are timbered where necessary, and the only difference so far, from the slicing method, is that the lower subs are developed more rapidly than the upper. In case the ore should turn out to be too soft for stoping, the system of mining could be easily changed into the slicing system. Commencing at points midway between the main raises, a series of chutes, with wide, flaring mouths, are put up to the bottom sub, and the miners begin to break half of the back between the bottom and second subs. The miners on the second sub then break the remainder of the bottom of that sub, and also

half of the back between them and the third sub. This is carried on up to the main level above. The lower subs are always drawn back a little ahead of those above, so that the broken ore will have a free drop to the chutes below, from which it is trammed to the shaft. (See sketch No. 6). The back of the main level is stoped down when it is no longer needed as a protection for tramming.

In the end, any system, suitable to the kind of ore, which may be adopted by a mine, must be altered more or less to suit a particular case. This is especially true of sub-level slicing. In some of the soft orebodies the ground does not crush as easily as in others and, as a result, the subs may be opened up farther in advance of the mining. The amount of a back which should be left over a sub, is a thing which must be worked out by experience for each mine.

In all of the mines of the district the aim is to increase safety in working conditions. Frequent inspections are made at all mines to see that the rules and regulations in regard to safe operation are carried out. That this policy has born fruit is evidenced by the decrease in serious and fatal accidents during the last half decade.

DISCUSSION.

MR. HARDENBURGH: Mr. Olsen, did you say your motors were provided with automatic gongs?

MR. OLSEN: Yes.

MR. HARDENBURGH: Were those put on there by the maker or added by you?

MR. OLSEN: I think in most cases they have been equipped at the different mines. I don't know of any manufacturers equipping their motors with automatic gongs. All of those that we have in use we have provided with automatic gongs in our own shops.

MR. HARDENBURGH: Those start to ring when the motor runs either way?

MR. OLSEN: Either way. They start ringing the moment the motor starts running.

MR. HARDENBURGH: What are they attached to?

MR. OLSEN: To the driving wheels.

MR. HARDENBURGH: To the axle of the motor?

MR. OLSEN: The device that we use consists of a small eccentric which is attached by a piece of strap iron to the motor frame and pressed against the flange of the driving

wheel with a spring. This spring keeps the pressure against the driving wheel at all times, so there is no possibility of the motor running without the gong ringing.

MR. SMALL: I would like to say that I know that the Goodman people furnish automatic gongs on their motors.

MR. BUSH: I would like to ask Mr. Olsen what the general practice on the Gogebic Range is now in top-slicing; whether they take the slice right up next to the top or leave a certain number of feet in the back to be taken as the miners pull back?

MR. OLSEN: Do you mean with reference to the back up over the sub?

MR. BUSH: Yes.

MR. OLSEN: The practice is the same as it was when you were here. We leave from 7 to 10 ft., according to the nature of the ground, above the back of the sub, and that is drawn as the sub is pulled back.

NEW STOCKPILE TRESTLE, COLBY IRON MINING COMPANY, BESSEMER, MICHIGAN.

BY G. S. BARBER, BESSEMER, MICH.*

The ore stocking trestle, in use at the Colby and Iron-ton mines, is different from that in general use in that it has bents of one leg instead of two, and is so designed to avoid some of the inconveniences of the common two-leg trestle. With the common trestle the beginning of steam shovel loading means to tear down stockpile trestle. The wiring for lights and motor, the rails and planking, stringers and caps are taken off and lowered to stockpile floor and hauled out of the way to be stored until the stockpiles are cleaned up, when they are again hauled back and the trestle is rebuilt with a loss of material and labor of two-thirds the cost of the original trestle. The trestle legs are pulled out of the pile when the shovel reaches them, or, as is often the case, are broken by a slide of ore, or with the shovel dipper and cut up for underground mining timber.

To avoid this loss and inconvenience the new trestle has bents of one leg only, spaced 32 ft. centers and guyed on each side, and stockpile is loaded without taking down trestle.

We claim for this trestle greater permanency, convenience in loading and stocking, and somewhat cheaper construction. Some of these trestles have been in use three years. They are the same in principle as the Negaunee mine concrete pier trestle, but are built of timber throughout, and while not absolutely permanent, are fairly so in that they do not have to be taken down every time stockpile is loaded and are much cheaper than concrete. In loading, the legs do not hold back the ore as do the two-leg bent; slides are fewer, and pulling out of legs necessary on two-leg trestle is avoided. There is also less hand shoveling. The legs being 32 ft. center to center, the steam shovel works well in between the bents and as shovel always works from the outside toward the center,

*Superintendent Colby Iron Mining Co.

what hand shoveling there is to do is on finer dirt than the outside rill of pile. Railroad tracks are laid along each side of pile, and after finishing one cut shovel is moved back on the loading track and started in on the other side of pile, al-



ONE-LEG TRESTLE, COLBY MINE. LOADING STOCKPILE



ONE-LEG TRESTLE, COLBY MINE. STOCKPILE CLEANED UP

lowing the first track to be moved in while the second cut is being made. With trestles 38 ft. high, three cuts clean up the pile.

The cost of construction, while not very much different,

is in favor of the one-leg trestle. We have found the difference about 25 cents per foot. With this system the trestle need not be taken down to load, but is always ready to stock even during the shipping season and this is often a convenience. Where stockpile room allows of two or more of these trestles, side by side, only one set of guys are needed on the



TOP OF TRESTLE SHOWING AMOUNT OF DISPLACEMENT AFTER ORE HAS BEEN LOADED.—COLBY MINE



THREE-TON TRAM CAR WITH MOTOR USED ON ONE-LEG TRESTLE.—COLBY MINE

outside, the trestles being guyed to each other on the inside.

These trestles have been described in the *Engineering & Mining Journal* of December 5, 1914, and *Excavating Engineer* of April, 1915. The design originated with Oscar Gustafson, Surface Foreman, at the Colby and Iron-ton mines.

Each bent is a single leg of 12 by 12-in. fir, 38 ft. long,

on which a 12 by 12-in. fir cap, 7 ft. long, is mounted and braced by two 6x8-in. by 6-ft. fir braces, mortised and bolted to both leg and cap. To each cap are bolted two 12x12-in. by 4-ft. fir corbels or bolsters, to which again are bolted the 8-x16-in. by 32-ft. fir stringers.

The stringers are trussed with 16-lb. rails; to each end of these a $\frac{5}{8}$ -in. plate is riveted and then bolted to the stringer, The truss rods are blocked in the center with a 6- by 12-in. wood piece. To the stringers are spiked 3-in. planks 5 ft. long, and the 30-lb. rails are laid on the planks at 30-in. gage. Outside of the 30-lb. rail, a 16-lb. guard rail is spiked.



STEAM SHOVEL WORKING ON LAST OR CLEAN-UP CUT. SHOVEL WORKING BETWEEN BENTS.

To each end of the cap is bolted a plate with an eye in the end, for attaching the guys. These guys are $\frac{3}{8}$ -in. galvanized-wire strands; they extend out to side bents erected at 100 ft. from the trestle, the guys from three center bents being attached to each side bent. The guys pass over the cap and down to eyebolts, passing through a 12-x12-in. by 16-ft. timber near the ground.

The side bents are 32 ft. high, built of round timber and well braced. They are themselves guyed by two $\frac{5}{8}$ -in. wire-rope guys to a "dead-man," concreted in the ground.

DISCUSSION.

MR. HEARDING: I would like to ask Mr. Barber the question as to whether in taking off the cut on the side you find that the pile on the other side commences to crowd over?

MR. BARBER: No, we haven't. After our stockpile has been loaded our tracks are in almost perfect alignment.

MR. HEARDING: I mean during the time you are taking off your first cut, does the pile on the opposite side crowd the trestle?

MR. BARBER: We haven't had any trouble. I think if it crowded over, it would remain crowded over and after the pile was cleaned up that would show. I don't know whether those cuts show that; I think they do. Yes, there is one cut that shows the trestle after the stockpile is loaded and the track is almost in perfect alignment.

MR. REIGART: Before you started using the single trestle and while you were using the trestle with the two legs, did you have any trouble with your trestle getting out of line and crowding over to one side or the other?

MR. BARBER: We always did. One leg would crowd out of alignment and it would throw our track a little side-wise. We would have to shim up our tracks.

MR. REIGART: This has been lessened by using the trestle with a single leg?

MR. BARBER: It has in our case.

MR. OLSON: What is the maximum load that is hauled over that trestle?

MR. BARBER: I don't know that I can tell the maximum load. We are sending out three tons of ore on a car, and we send two cars to a locomotive. It is a four-ton locomotive, and the cars probably weigh three-quarters of a ton, which would be $1\frac{1}{2}$ tons for the two. I would say that it was possibly $11\frac{1}{2}$ or 12 tons. These bents are 32 ft. centers, with stringers 8 by 16 and trussed with 16-lb. rail.

MR. RICHARDS: What do you consider the life of one of those trestles?

MR. BARBER: I could only make a guess and you could do the same. I don't know. They pay for themselves, I think, after you have loaded your stockpile the second time; I think you have got the worth of your money. They cost less than the trestle with two legs and they are more satisfactory.

MR. RICHARDS: Do you put a concrete base under the posts?

MR. BARBER: No, we probably should, but we don't and we have had no trouble. The stockpile floor is well packed.

MR. RICHARDS: Are these posts set in the ground?

MR. BARBER: They are set on a square piece of timber, sometimes the planking of the stockpile floor.

MR. BUSH: I would like to ask Mr. Barber if he cares to give us any comparative cost of loading as between the two different styles of trestle?

MR. BARBER: So many conditions enter into that, I don't think that any figure I could give would be of much value.

MR. BUSH: It seems to me that having permanent tracks, and not having to pull out the legs, the cost per ton of loading would be very much lower?

MR. BARBER: That varies so much. It depends on whether we have continuous car service or otherwise. In some cases our loads have to be switched out and placed a long ways from our stockpile floor. In that case we can only load about half the time. Those things depend so much on conditions that the figures would not be of much value.

MR. BAXTER: You take a cut on one side and then swing over and take a cut on the other side?

MR. BARBER: Yes.

MR. BAXTER: I was thinking that the posts would bar the swing of the boom so that you couldn't work up very close and that you might leave quite a little ore?

MR. BARBER: You may have noticed today, if you went over the stockpile floor, what we leave without hand shoveling; there is a little pile around the legs, not over 6 ft. in diameter.

MR. HEARDING: Do you start filling from the shaft-house and only put your tram car and not your motor on the empty trestle? In other words you don't put your cars or your motor out onto that trestle before you have filled up around the foot of the trestle?

MR. BARBER: Yes, we do, but we don't fill the farther end first. We fill the trestle gradually. We work along the full length of the trestle—not the full length of the trestle perhaps, but for a considerable distance. We don't fill up a pile right close to the shaft-house and then continue out. We generally put it all the way out for about 200 ft. and then gradually fill that up. If we knew exactly what we were

going to stock on that trestle, we would make it just long enough to accommodate that amount. We find that it doesn't crowd the legs.

MR. KELLY: Is there more than one grade of ore?

MR. BARBER: No, but we have loaded two grades of ore on the same trestle, one at one end and one at the other.

GROUTING AT THE FRANCIS MINE SHAFT OF THE CLEVELAND-CLIFFS IRON COMPANY.

BY J. R. REIGART, PRINCETON, MICH.*

The Francis Shaft was sunk through quicksand to ledge a distance of 102 ft. by The New York Foundation Company, who completed the work in June, 1910. As we were at that time not ready to continue the shaft The Foundation Company put a concrete seal in the bottom and turned it over to the company.

The shaft through the sand is circular, 17 ft. in diameter, inside dimension. It was constructed so that it might be continued in the standard size adopted by The Cleveland-Cliffs Iron Company—10 ft. 10 in. by 14 ft. 10 in. inside measurements. This standard shaft is rectangular with two skip-compartments, a cage-compartment, and a ladder- and pipe-way. Work was resumed in the spring of 1911, but continued only for a short time, just long enough to put in the steel dividings in the concrete shaft, and to drill a number of holes through the seal in the bottom of the shaft to ascertain the flow of water to be handled. The holes put down all struck water at a depth of from 2 to 3 feet. The water came out under a pressure of a little over 40 pounds, and any two of these holes made sufficient water to give a No. 9 Cameron pump just about all it could handle. As fast as the holes were drilled, wooden plugs were driven into them to shut off the water. At first any pieces of wood at hand were used as plugs, but these did not entirely keep back the water. Accordingly regular soft-pine plugs, from 3 to 3½ ft. long and tapering from 2 to 4 in., were turned out in the shops. With these the holes could be made water-tight.

The holes drilled demonstrated the fact that a large flow of water would be encountered as soon as the seal was broken, and also that the slate ledge was probably broken to a greater or less degree. It was readily seen that the water would

*Assistant Superintendent.

have to be excluded before continuing the shaft, or the shaft could be sunk only with the greatest difficulty and expense, if at all. The Foundation Company might have carried the shaft down farther by their own process except for the fact that at the time ledge was struck the men were working under 47 pounds air pressure on 15 minute shifts. This was not only exceedingly expensive, but hazardous.

The ledge at the bottom of the shaft being quite irregular, the thickness of the seal varied, but it was supposed to average 24 inches. The encountering of water so quickly in the holes seemed to indicate that the seal put in by The Foundation Company had not formed a good contact with the ledge. Just at this point work was discontinued and was not resumed until the first of February, 1915.

When work was begun anew last February various means were discussed for permanently cutting off the water which would come in at the ledge, and it was finally decided to attempt this by drilling incline holes around the inside circumference of the shaft, at such an angle that they would reach beyond the outside circumference of the wall of the shaft, and forcing neat cement into these holes under air pressure until all the water-bearing cracks and crevices were filled, thus making a water-tight ledge through which the shaft could be sunk with safety. As stated above, we were afraid that there was an open space between the seal and the ledge covering a good portion of the area of the shaft. If this was the case, the pressure exerted on the cement to force it into the holes would in turn be transmitted against the bottom of the concrete seal and develop an enormous pressure. Thus there would be great danger of breaking through it, and breaking through would be a very serious matter. As a precaution against this, 3-in. planks were set up on edge 8 in. apart, like joists in a floor, and on top of these and at right angles to them 12-in. square timbers were put in across the shaft about four feet apart and spragged with stulls to the steel sets above. Wherever there was any space between the joists and the cement seal, wedges were driven in so that should there be any tendency for the concrete to give, the pressure would be instantly transmitted to the bracing. This covered up the bottom of the shaft pretty well, but still left space enough for drilling holes until it was found that the space below the seal had been filled and the reinforcing could be removed with safety.

Six feet above the bottom of the shaft a platform was built, on which was installed the air-pressure grout machine, the valves for operating the low- and high-pressure air line, and a No. 9 Cameron pump. On the next set above, two other pumps were installed, a No. 8 Cameron and an Alberger electric pump. This gave a total pumping capacity of from 1400 to 1600 gallons a minute. A 4-in. pipe line for conveying the grout, which was to be mixed on surface was installed from surface so that it emptied directly into the grout pan. A mixing tub made by sawing an oil barrel in half was



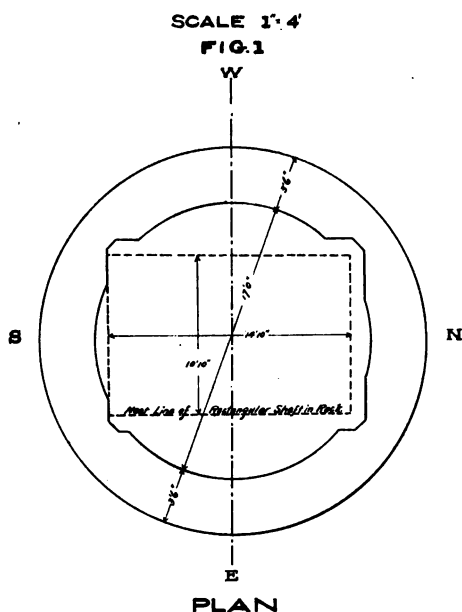
PLATE 1. SHOWING GROUT MACHINE AND CONNECTIONS. FOUR INCH SUPPLY PIPE FROM SURFACE EMPTYING INTO TANK.

placed over this pipe. A few feet from the mixing tub a cement house was erected and water piped to it. Two air lines were set up in the shaft, one from the compressor which furnishes the mine for ordinary conditions at 80 pounds and a 6-in. high-pressure line from the booster installed in the engine house. This high-pressure line also acted as a receiver. The booster was a $9\frac{1}{2} \times 9\frac{1}{2} \times 10$ -in. Westinghouse Air-Brake compressor, such as is used on locomotives. It is run by steam and is capable of raising the low pressure up to 250 pounds.

The machine used on this job was a second-hand one which had been repaired and 150 pounds pressure was about the most we could get out of it. Stopcocks were arranged so that when desired the low-pressure air could be quickly shut off and high-pressure air forced into the grout tank instead.

The tank used was the Caniff Grout Machine. It consists essentially of an iron tank of approximately 24x48 in. with connections for allowing air to be blown in at both the top and bottom. When the tank has been charged, air is blown in

FRANCIS MINE SHAFT



SHOWING SHAFT DIMENSIONS

at the bottom to agitate the grout, which insures it a thorough mixing and keeps it from setting. This air is then shut off and air let in at the top to force out the grout. A stopcock with a long handle controls the flow of grout through the discharge pipe. The trapdoor at the top through which the charge is inserted is held shut when closed by the air pressure. After the charge has been expelled from the tank and all the connections, the air from the air line is shut off from the tank, the chamber exhausted by means of a relief valve on the top of the machine, and the charging door opened

for a fresh mixture. These tanks come equipped with valves, but stopcocks were substituted for them, as they can be worked much more easily and quickly. The grout pan should be set up as near the working place as possible, as all parts of the

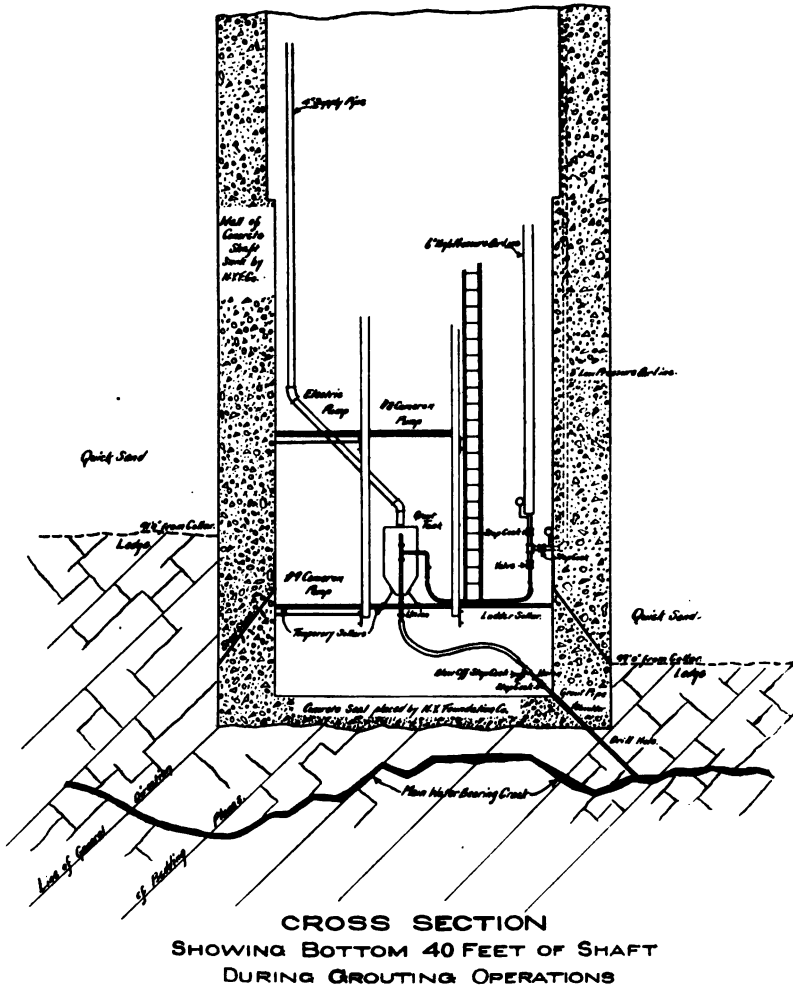
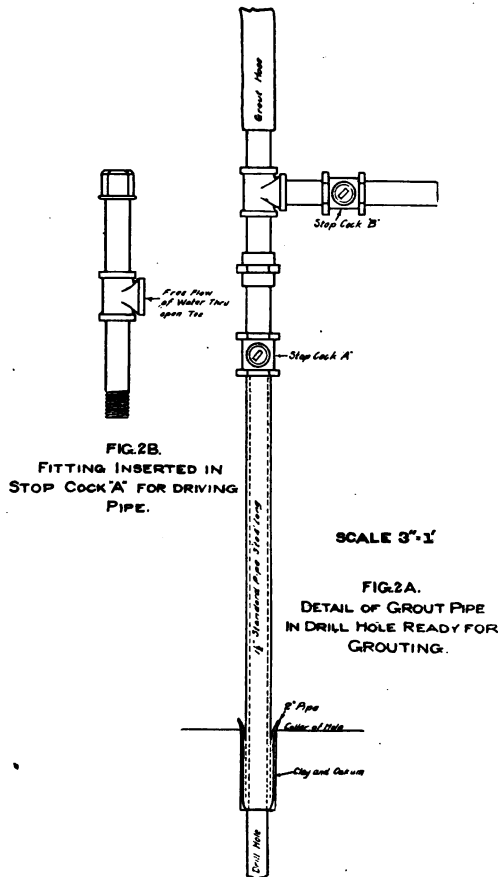


FIGURE 1

outfit are thus much more easily controlled, and especially as it shortens the connection between the grout tank and the hole being grouted and prevents clogging. The cross-section

in Fig. 1 shows the arrangement of equipment described above and Plate 1 illustrates the grout machine and its connections.

The connection between the tank and the grout pipe was a 2-in. rubber hose of extra quality warranted to stand 300 pounds pressure. One end was fitted up with a 1½-in. half-union for connecting it to the discharge pipe of the grout machine, and the other end was supplied with a blow-off valve and



a 1½-in. half-union for coupling it to the grout pipe. (See Fig. 2-A). The blow-off valve (Stopcock "B") served in finding out whether or not all of the charge had gone into the hole, in emptying the tank of air after the charge had been ejected, and in blowing out the hose after the hole would take

no more grout. The function of emptying the tank of air after the contents had been discharged is quite important if speed is desired, as the relief valve on the tank for this purpose freezes up if worked too rapidly. The hose with connections was about 14 ft. long, sufficient to reach any point in the bottom of the shaft and yet not long enough to make loops or troublesome unnecessary curves.

The grout pipes were made from $1\frac{1}{2}$ -in. standard pipe

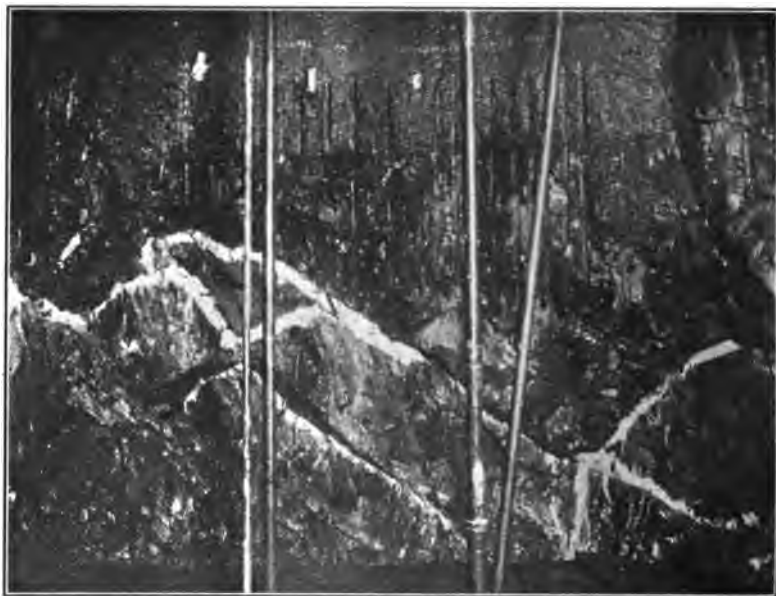


PLATE 2. SHOWING WATER BEARING CRACKS WHICH HAVE BEEN FILLED WITH GROUT. THE CLEAN-CUT SHARPLY DEFINED WHITE SEAMS SHOW THE EFFECT OF LETTING THE SAND OUT SO THAT THE SPACE IS FILLED WITH SOLID CEMENT. THE BLURRED, LESS DISTINCT SEAMS, HAVE FINE SAND MIXED WITH THE GROUT AND WHILE WATERTIGHT THE FILLING MATERIAL IS NOT SO HARD AS THE CLEAN GROUT.

3 to 4 ft. long. To the end to be driven into the hole, a bell made from a piece of 2-in. pipe 8 in. long was welded. The $1\frac{1}{2}$ -in. pipe is pushed into the large end of the bell until its end is even with the unexpanded end of the bell and welded at this point. (See Fig. 2-A). The other end of the pipe was equipped with $1\frac{1}{2}$ -in. stopcock, a $1\frac{1}{2}$ -in. nipple, a $1\frac{1}{2}$ -in. tee, which was left open on the side to permit the free flow of water while the pipe was being driven, and a nipple

on to which was screwed a cap. (See Fig. 2-B). In drilling the holes, a starting bit of $2\frac{3}{4}$ in. was used to a depth of 8 or 10 inches. The next bit was $2\frac{1}{2}$, but the succeeding bits followed the usual change. The difference in size between the starter and the second drill provided a shoulder



PLATE 3. SHOWING GROUT HOSE CONNECTED UP TO PIPE READY FOR GROUTING A SMALL STREAM, ABOUT 5 GALLONS PER MINUTE, NOT SHUT OFF BY ORIGINAL GROUTING. ABOVE ARE SHOWN ORIGINAL GROUT PIPES, NOW CUT OFF TO ALLOW THE SHAFT TO BE DRILLED. MARKS OF THE HOLES PUT DOWN TO TAKE OUT THE FIRST ROCK CUTS ARE ALSO CLEARLY SHOWN ON THIS PLATE AND PLATE 2.

against which the grout pipes could be tightly driven. Before the pipe was driven into the hole, the bell was wrapped with oakum and this in turn was coated with clay. The pipes were driven with a sledge, a wooden block being held on the cap to receive the blow. After the pipe was driven as far as

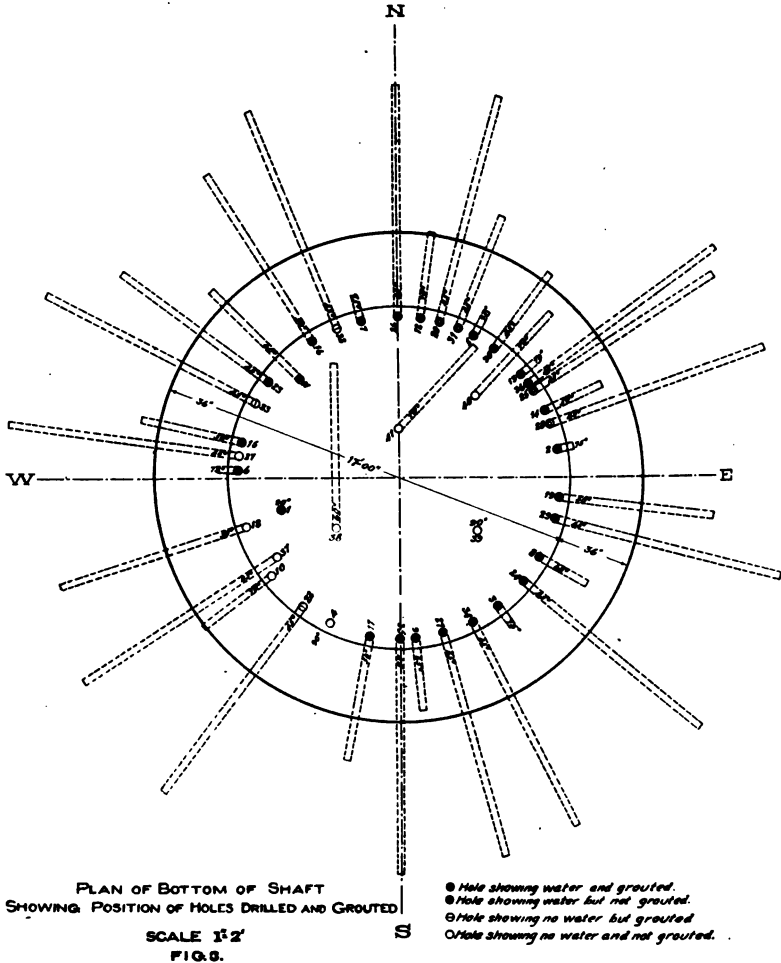
it would go, the stopcock ("A" Fig. 2-A) was closed, and if there was no leakage around the pipe at the collar of the hole, the nipple in the stopcock, with its attached tee and cap, was replaced by another nipple to which was attached a half-union for the coupling with the grout hose.

If there was a little leakage, it was stopped by driving small steel wedges, shaped to conform to the pipe, between the pipe and the upper end of the bell welded to it. This upper end of the bell was drawn to a feather edge, which conformed to any little irregularity in the shape of the hole, and which extended out over the Oakum and clay and prevented it from being forced up on the pipe. The effect of the steel wedges was to force back this feather edge and make a tighter contact against the collar of the hole. If this did not stop the water, small soft-pine wedges were driven in at the leaks. There were times, however, when even with wedging the pipes could not be made tight the first time and had to be taken out and put in until they were tight. However, a few wedges around the pipes was usually all that was required. The drawings 2-A and 2-B and the photographs illustrate the details here described.

The general routine followed is illustrated by Fig. 3. A unit of operations was the drilling and grouting of six holes spaced at equal intervals around the circumference of the shaft. The holes were numbered in the order drilled. To drill these six holes and put in the grout pipes was a days' work for a crew of 2 men and a mining captain. On the following day these holes were grouted, and for the next two work was suspended to give the grout time to set up and harden. At first only one day was allowed for this, but it proved insufficient in cases where a large amount of grout had been put into one hole. A total of 41 holes were drilled and 27 were grouted. As the work progressed the holes were drilled deeper, until 14-ft. holes put in at an angle of 45° encountered no water. Finally a few holes, Nos. 38, 39, 40 and 41, were put down in the interior of the shaft to reach any seam that might run up into the shaft parallel or nearly parallel to the incline holes which these holes might have missed. As no water was encountered, all was now ready for the excavating of the bottom of the shaft and for shaft-sinking.

Grouting was started on the morning of March 26. A gauge attached to one of the grout pipes showed a pressure

of 43 pounds. To begin with, it was surmised that there were relatively large areas to be filled, so that the only pressure to be overcome would be the hydrostatic head. Sixty pounds was therefore used in the first operations. The first mixture was very thin, being composed of two 12-qt. pails



of cement and four 12-qt. pails of water, and as a result, the grouting went very slowly. Therefore the mixture was changed to one bag of cement and three 12-qt. pails of water to a batch. It is most important, however, that the mixture be thin enough so that it is perfectly fluid. No sand was

used in these mixtures, but it may be used if desired. All of the cement was screened over a heavy wire screen running four meshes to the inch, to take out any lumps. The first hole grouted was one of the holes drilled in the operations of 1911. Some of the plugs put in at that time were not tight enough to keep the thin mixture of grout from being forced back up into the shaft. Wherever possible, these old plugs were replaced by plugs previously described. This helped a great deal, but there were some plugs which could not be removed and which could not be made tight enough to hold back the grout. Accordingly several batches of bran were forced into the hole, and then a richer mixture made with one bag of cement to a batch. The holes soon took up and there was no more trouble from this source.

From the first holes in particular, the water brought up fine sand and small pieces of broken ledge. With the very first of these, the water was shut off as soon as the grout pipe had been made tight, but later the stopcock was left open and the water allowed to flow for a period of ten minutes, until the water became clean and free of sand. The effect of this was seen later when the bottom of the shaft was cut out: in holes which had not been allowed to clear themselves the grout was mixed with the sand in the seams and could be readily taken out of the cracks with a pick, but wherever the holes had been allowed to clear themselves the cracks were filled with hard, clean cement. The table following gives the record of each of the holes:

RECORD OF HOLES DRILLED AND BLASTED.

No. of Hole	Depth Ft.	Inclination In. degrees	Cement Bags	Flow gal. per minute	Pressure lbs.	Water	Remarks
1	3	90	18	250	55	Sandy	
2	2 8	75	16	50	55	Clear	
3	4	75	6	200	60	Sandy	
4	5 6	90	0	0	60		
5	4 6	75	150	250	60	Sandy	
6	3 2	75	190	250	60	Sandy	24 qt. cement and 48 qt. water
7	4 1	75	4	25	60	Clear	
8	5 11	68	2	75	60	Sandy	
9	7	65	29	75		Sandy	
10	10	70	0	0			At a depth of 3 or 4 ft., went through grouted seam about 3 or 4 in. thick

No. of Hole	Depth Ft.	Inclination In. degrees	Cement Bags	Flow gal. per minute	Pressure lbs.	Water	Remarks
11	10	2	60	0	75	Sandy	
12	10	4	70	40	75	Sandy	
13	3	4	70	332	100	Sandy	
14	4		50	6	50	70-75	Sandy
15	6	6	50	2	50	70-75	Sandy
16	12	6	50	2	25	70-75	Clear
17	8		50	7	75	70-75	Clear
18	12	6	50	0	0	70-75	
19	10		50	115	100	70-75	Sandy
20	13	6	45	0	2	140	Clear
21	13	6	45	5	10	140	Clear
22	13	6	45	1	0	140	Clear
23	10	6	45	39	50	75	Clear
24	13	4	45	2	15	140	Sandy
25	12	6	45	0	50		Clear
26	13	4	45	2	1	140	Clear
27	13	6	45	0	0		
28	13	3	45	0	75	Sandy	Stopped from No. 30
29	13	6	45	4	50	130	Sandy
30	5	6	45	257	150	80-140	Sandy
31	7		45	0	25		Clear
32	13	6	45	2	0	140	
33	13	7	45	2	0	140	
34	13	4	45	3	15	120	Clear
35	13	8	45	2	7	140	Clear
36	13	7	45	1	1	140	Clear
37	13	5	45	0	0		
38	13	6	60	0	0		
39	13	6	90	0	0		
40	13	6	70	0	0		
41	13	6	70	0	0		
Tot. 405			4	1239			

Grouting began March 26, 1915. Finished, April 26, 1915.

On any day's grouting, the hole showing the largest flow was generally the first one connected to and grout was forced into it as long as it could be made to take any. When once started, the grouting of a hole was finished without any stop, for if the cement was allowed to start to set, no more could be forced in and the hole would be lost. Hole No. 13 required 332 bags of cement; grouting was started at 8:30 and run continuously until 3 p. m. Towards the end of the operation the holes took grout more slowly and if at any time the grout hose did not empty after the pressure had been left on for five minutes, the hole was considered finished. Sometimes in such cases the hose would have to be hammered and high-pressure air applied to it to clean it out. After a couple of days' work with 60 pounds of air, it was decided that the seal and bracing were strong enough to stand a pressure of 80 pounds, and from then on this pressure was used almost

entirely. Some holes that would not take grout at 80 pounds were made to take a few batches under a pressure of 150 pounds. This pressure was also used towards the completion of the holes when they began to take grout slowly at 80 pounds. When the holes were taking grout freely we operated at the rate of a batch every 50 seconds, and this was just about as fast as the cement could be screened and mixed and sent down to us from surface. The grouting crew on surface consisted of five men, one man bringing the cement from the cement house, one man screening, one measuring out the charges, and two mixing them and stirring them with paddles to insure a thorough mixture. In the shaft the captain operated the grout machine, one man the blow-off valve near the grout pipe, and one the stopcock on the grout pipe.

At the end of a day's grouting, the stopcocks on the pipes used for the last previous grouting were opened. If no water came out, the stopcocks were taken off and thoroughly cleaned, so that they could be used over again. If water flowed from the pipes the grout machine was connected to the grout pipe and a batch of clear water forced into the hole. If the hole took the water, as many batches of grout were forced in as the hole would take. This amounted to anywhere from one-half a batch to two or three batches, and resulted in cutting the water off entirely. High pressure was used for these finishing batches.

Where the openings to be filled are as large as they were in this case, 80 pounds of pressure will make a good job, but where the seams are smaller, higher pressure is not only desirable but absolutely necessary in order to overcome, in addition to the hydrostatic head, the friction in the seams being filled, and the tendency of the previous batches to set up. The farther in the grout can be forced, the larger will be the intervening wall built up and the more effectively will the water be cut off. If high pressure could have been used from the beginning, this particular job could have been done with fewer holes and grouting operations. The grouting was finished on April 26, a month after starting.

When it had been thoroughly demonstrated that the water had been all cut off, the rectangular shaft was started. Eight-foot holes were drilled around the perimeter of the rectangle two feet back from the neat line of the shaft and as close together as possible. The first four feet were then broken out by moiling and with wedges and feathers. The concrete seal

was the hardest part of this work, as the ledge broke easily along the slips and joints. Below these four feet shallow holes and very light charges of powder were used. For the next cut the holes were drilled on the outside of the shaft as before and the ground broken out by drilling and blasting with light charges of powder, not over half a stick to a hole, and never over two or three holes at a time. This operation was repeated until the shaft was 18 ft. below the shoe of the concrete shaft sunk by The Foundation Company.

To secure further the portion of the shaft which had been grouted, the sides of the shaft traversed by the filled seams were lined with a reinforced concrete wall averaging two feet in thickness. This wall required 14 ft. of concrete lining. Hitches were cut 12 ft. below the shoe and 9-in. I-beams put in for bearers along the small dimension of the shaft. To these bearers, hanging bolts were attached so that the sets could be hung below. Forms were then constructed along the neat line of the shaft from the top of the concrete seal to 2 ft. below the bearers. First, however, 1 $\frac{3}{4}$ -in. cramp rods spaced 2 ft. apart were put in the walls of the shaft. Wire rope, old pieces of expanded metal, iron bars and pieces of old angle iron and channels were fastened in for reinforcing. The space was then filled with concrete having a composition of one part cement, one part sand and gravel and two parts broken rock. This was given a couple of days in which to set and then work was resumed. For the first two or three cuts the perimeter was drilled around as in the previous cuts and the holes blasted carefully so as not to damage the concrete.

When the bottom of the shaft was taken up, the conditions were not exactly as had been anticipated, as the seal had a good contact with the ledge over the entire area of the bottom of the shaft. What we did find, however, was a large seam varying from 2 to 6 in. in width cutting across the entire shaft. On the south and east side it was within a few inches of the bottom of the seal while on the north and west side it was farther down in the shaft. In the southwest corner it apparently made a turn and went outside of the shaft, as no water was encountered in any of the holes—No. 4, 10, 18, 22 and 37—put down in this area. This seam filled with cement is shown in Plates 2 and 3. From six places in the sides of the shaft small streams of water issued, the combined flow of which was not over 25 gallons per minute. Holes were drilled

into the rock at these points and plugged with grout pipes so that later they could be grouted and the water stopped. In addition several weep pipes were put in, which all ran dry as soon as the concrete set.

After the 14 ft. of concrete had been allowed to set for six weeks, the sides of the shaft were well braced and the holes making water and the weep pipes grouted. The bracing of the shaft was to prevent the sides from bulging and breaking should there be any openings between the concrete wall and the rock sides of the shaft—making areas against which the air pressure used in grouting might be transmitted. Although some of the water was shut off and the total flow cut down to about 15 gallons per minute, it could not be cut off entirely with the available air pressure. The openings through which the water came were evidently so fine that only a higher pressure would force the grout into them. During the time between the installation of the concrete lining and the last grouting, the shaft was sunk 16 feet. After the weep holes were grouted the forms were removed, dividings installed and the shaft equipped for sinking in the ordinary way. Up to this time all the work had been done on day shift only.

To reach this stage four months were required, from March 26 to July 27. The greater portion of the last three months was spent in sinking, which, because of the care which had to be exercised, was slow and tedious.

The cost of this work up to the first of August, when two shifts could be employed and sinking carried on in the usual way, was as follows:

Grouting	\$1,431.74
Sinking	1,882.45
Timbering	885.15
Concreting	407.42
Total	<u>\$4,606.76</u>

When the advantages gained are taken into consideration, the method used seems very inexpensive. The actual flow of water there would have been to handle if it had not been shut off is entirely problematical, but 4,000 to 5,000 gallons of water per minute is undoubtedly a low estimate. Suppose the shaft could have been sunk to the present depth by ordinary methods and that the water could have all been caught and pumped from ledge—the expense of installing the necessary

pumping equipment and of pumping this amount of water over the period of years comprising the life of the mine would be enormous. It is very obvious that one could afford if necessary to spend several times the above amount to get the results obtained.

The method of grouting water-bearing seams can be used at any depth provided sufficient air pressure is available. For real effective results, the air pressure should be from 100 to 200 pounds over the hydrostatic head.

No further trouble from water is anticipated in the sinking of the Francis Shaft, as the drill hole put down on the shaft location to a depth of 865 feet was found in the bottom of the shaft and no water was coming from it.

SHEET GROUND MINING IN THE JOPLIN DISTRICT, MISSOURI.

BY EDWIN HIGGINS.*

The Joplin zinc-lead district, also referred to as the Missouri-Kansas-Oklahoma district, includes portions of Jasper, Newton, Lawrence and Greene counties, Missouri; Cherokee county, Kansas; and Ottawa county, Oklahoma. The bulk of the production of zinc and lead concentrates, however, comes from the mines in the vicinity of Webb City, Cartersville and Joplin, in Jasper county, Missouri. In recent years, the annual value of the zinc and lead concentrates produced in the entire district has ranged from 11 to 18 millions of dollars, from 82 to 85 per cent. of this being derived from the zinc. There are approximately 10,000 men employed in the district.

With regard to mining methods, the ore deposits of the Joplin district may be divided into two classes: (a) Sheet ground deposits; flat-lying orebodies in tough, bedded, cherty flint; and (b) "Runs" and irregular deposits; orebodies of a great variety of shapes and forms, in which the mineral occurs in the chert or in the dolomitic limestone.

In past years much mining was done at or near the surface and, while this is true at the present time to a certain extent, the bulk of the production comes from depths of from 150 to 200 feet. The deepest mining operation of the district is at Miami, Oklahoma, where a pump station was cut recently on the 380-foot level of the Lennan mine.

The principal sheet ground mines are situated in the vicinity of Joplin, Oranogo, Webb City, Cartersville, Prosperity, Porto Rico and Duenweg, in Jasper county, Missouri. At this time there are in operation about 60 sheet ground mines, employing close to 5,000 men. The recent unprecedented demand for zinc ore has been the cause of increased activity here, as in other parts of the district.

* Mining Engineer, U. S. Bureau of Mines, Ironwood, Mich.

SHEET GROUND OREBODIES.

The geology of the Joplin district has been treated exhaustively in various publications*. Only such comments will be made here as may seem necessary for an understanding of conditions in their relation to mining methods.

The principal sheet ground orebodies lie at an average depth of 180 feet from the surface. They occur in the Grand Falls member of the Boone chert. This formation belongs to the Mississippian series of the Carboniferous system. The chert, or cherty flint, as it seems more properly termed, occurs in layers from a few inches to three feet in thickness, these layers comprising beds varying in thickness from 10 to 40 feet or more; dolomitic limestone lies above and below these approximately horizontal strata. This flint is very tough and breaks in various splintery and sharp-edged forms. It ranges in color from almost white to gray or bluish gray. In places it is unaltered; in others it has been crushed in place and recemented with a siliceous material, the bed as a whole retaining its original form. In parts of the area many fissures are found in the flint.

The following is a record of a typical drill hole put down in the sheet ground:

	Feet.
Soil	2
Yellow clay	18
Gravel	10
Yellow clay	10
Soapstone	64
Alternate layers of chert and dolomitic limestone	116

*Schmidt, A., and Leonhard, A., Lead and Zinc Regions of Southwestern Missouri; Mo. Geol. Surv. Vol. 1, 1874, pp. 381-502.

Winslow, A., Lead and Zinc Deposits, Mo. Geol. Surv. Vols. 6, 7, 1894.

Jenny, W. P., Lead and Zinc Deposits of the Mississippi Valley; Trans. Am. Inst. Min. Engrs., Vol. 22, 1894, pp. 171-225; 642-646.

Bain, H. F., Van Hise, C. R., Adams, G. I., Prelim. Report on the Lead and Zinc Deposits of the Ozark Region: 22nd An. Report U. S. Geol. Surv., part 2, 1901, pp. 23-228.

Smith, W. S. Tangler, Lead and Zinc Deposits of the Joplin District; Missouri-Kansas; Bulletin U. S. Geol. Surv. No. 213, 1903, pp. 197-204.

Buckley, E. R., Buhler, H. A., Geology of the Granby Area; Missouri Bureau of Geology and Mines; 2nd series, Vol. 4, 1906.

Haworth, E., Crane, W. R., Rogers, A. F., and others. Special Report on Lead and Zinc; Univ. Geol. Surv., Kansas, Vol. 7, 1904.

True sheet ground (chert) 45
Dolomitic limestone to bottom of hole at....290

It may be noted that there was in this hole 45 feet of sheet ground. The depth at which the sheet ground lies, its thickness, and the number of beds that may be encountered, varies in different parts of the area. It may lie anywhere from 100 to 240 feet below the surface, and it may be divided into two or more beds, with limestone between.

The minerals which form the orebodies are sphalerite, or zinc sulphide (locally termed "jack"); and galena, or lead sulphide (termed "lead"). These minerals usually occur closely associated, and in widely varying proportions, in well-defined bands from a fraction of an inch to 6 inches in thickness. These bands lie between the bedding planes of the flint and vary in frequency. The entire orebody may consist only of one band of mineral, or it may be made up of two or more bands at varying distances apart.

Obviously, the height of the mine workings is dependent on the frequency of the mineral bands. If the flint is barren, with the exception of one mineral band, only about 6 or 7 feet are mined. In some parts of the district the mineralization warrants the mining of 20 and even 30 feet in thickness.

Under normal conditions, as to cost of labor and price of ore, there is little or no profit to be had when the "dirt" runs below 2 per cent. "Two per cent. dirt" means rock that yields 2 tons of concentrates for every 100 tons mined. Broken rock in the mine is termed "dirt"; the concentrates are termed "ore." In this connection it might be well to mention that approximately 30 per cent. of the mineral content of the dirt goes into the mill tailings pile.

PROSPECTING AND DEVELOPMENT WORK.

Churn Drilling—Owing to the evenness of the surface and the presence of soil, clay, sand and boulders to varying depths, it is seldom that any information can be obtained from rock outcrops. Hence practically all prospecting is done by means of the drill. For this work Keystone and Star churn drills are largely used. The usual practice is to start a hole with 6-inch and end it with 4-inch casing. Generally a 200-foot hole is all that is necessary to determine the presence or absence of pay dirt in the sheet ground district. The cost of

drilling ranges from 90c. to \$1.00 per foot. The average time required to put down a 200-foot hole is two weeks.

It is worthy of note that this form of prospecting and locating an orebody is very cheap in comparison with methods required in other metal mining districts. For instance, a total of 25 holes 200 feet in depth will serve to prove up a large acreage; or, placed close together, will define an orebody of considerable extent. This number of holes would not cost over \$5,000.

Outline of Development—After the orebody has been defined by drilling, a shaft is sunk, and the cutting of the station begun. Pillars are left to protect the shaft and mining begins, more machines being put in use as the mine workings extend. The "ground" (rock) is drilled and blasted down and the resulting "dirt" (broken rock) is shoveled into "cans" (buckets). The cans are trammed singly by hand, or in trips by power or mules, to the shaft station. Here they are hooked on to the cable, hoisted to the top of the "derrick" (headframe) and dumped onto a 5- or 6-inch grizzly. From here the dirt starts on its way through the mill. This in general is the method followed in all of the mines.

Shafts—While there are a few two-compartment shafts in the district, it is customary to sink a single-compartment shaft from 4x5 to 5x7 feet in section inside timbers. Corner, side and sump holes are used in sinking, only one set of the latter if the ground is soft. Timbering is required only in the soft ground above the limestone. The usual method of timbering a shaft is to place 2x4's, without framing, in crib fashion around the four sides, the spaces left between the 2x4's being filled by shorter pieces of the same material. The only blocking used consists of 2x4's placed vertically against the outside of the cribbing, and extending from the shaft collar to the bottom of the timbering. Two of these vertical supports are used for each side of the shaft.

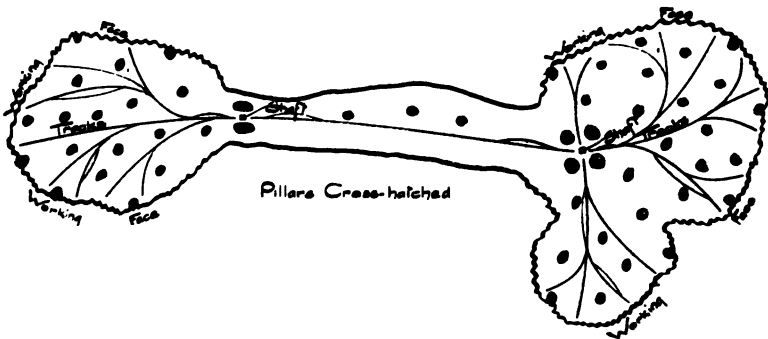
There are no ladders in the shafts, men being handled in the buckets, and supplies and machinery either in the bucket or from the cable. Air lines, water columns and electric power lines (where used) are placed in a corner of the shaft. No crosshead or guides are used, the bucket being allowed to swing freely in its passage through the shaft.

The Shaft Station—Usually no definite plan is followed in the arrangement of the shaft station. After the sinking of the shaft development proceeds in the direction of the ore-

body. This may lead only in one direction, or it may extend on all sides. As mining progresses there is usually ample room for any arrangement of tracks that might be desired.

MINING METHODS.

To better understand the methods of mining it may be well first to get an idea of the appearance of a working mine. Sketch 1 shows in plan a typical sheet ground mine. The workings consist of an approximately horizontal excavation, the height depending on the thickness of the pay dirt, the lateral dimensions being governed entirely by the extent of the orebody. The continuity of this flat-lying excavation is broken only by the presence of pillars, left to support the roof and overlying strata. A miniature model of one of the mines



SKETCH 1. PLAN OF SHEET GROUND MINE

may be constructed by simply placing a number of ordinary spools upon a table, from 4 to 5 inches apart, and placing a book on top of the spools. The table will represent the floor of the mine; the spools, the pillars; the open space, the worked out portion; and the book, the roof.

Carrying the Face—An idea of how the face is carried may be obtained from Sketches 1 and 2, the latter being enlarged to show how the holes are placed and the pillars blocked out. Leaving out of consideration the method of breaking the ground, the term "longwall advancing" perhaps best describes the manner of carrying the face.

The method of breaking the ground is governed largely by the height, or thickness, of the pay dirt; and to a certain extent by the nature of the ground. Up to a height of 15 or

even 18 feet the face is carried vertically, the ground from floor to roof being broken by one round of shots. This will be termed "mining in the heading." Where the deposit is from 25 to 30 feet thick a bench, or stope, and a heading are carried.

It happens frequently that after a mine has been supposedly worked out, say 8 or 10 feet in thickness, subsequent prospecting develops the fact that there is payable rock in the floor. To mine this lower stratum it is only necessary, after cutting out the floor at the shaft station, to drill long stope holes, or lifters. This is called taking up stope and is a very cheap method of mining, one machine doing the work of 6 to 8 machines in the heading. Also, the powder cost is much lower.

Breaking Ground in the Heading—A large proportion of the mining in the sheet ground is in deposits from 7 to 18 feet thick, the face being carried vertically as shown in Sketch 3, Fig. A. By reference to Sketch 2 it may be noted that a plan of the working face presents an irregular outline. Obviously, breaking to the irregular faces thus exposed is easier than shooting from a solid face. For convenience these irregular faces will be termed "sub-faces."

The number and location of the drill holes for a given round depends chiefly on the shape of the block of ground to be blasted. Anywhere from 3 to 6 holes may make up a round. As indicated in Sketch 2, all holes are drilled approximately parallel to the sub-face. The dotted lines show the ground that will be broken by the round of shots. Sketch 4 shows three common methods of placing holes. The numbers indicate the order in which the holes are fired. In Fig. A, hole No. 1 is termed the relief, No. 2 the front breast, No. 3 the front stope, No. 4 the back roof, No. 5 the back breast, and No. 6 the back stope. Heading holes range from 6 to 15 feet in length, depending on the height of roof. The length of hole corresponds roughly to the height of roof.

Drilling—The bulk of the drilling is done with solid-steel piston drills, operated with air at pressures varying from 60 to 90 lbs. at the machine. Recently, in two or three of the mines, there have been tried, and with success, various types of hammer drills using hollow steel through which water passes to the back of the drill hole.

For working in the heading the machine is mounted on a column. A compilation of records of heading work in the sheet ground (7 to 18 feet in thickness) shows that one ma-

chine will drill from 20 to 35 feet per shift, and that the production per machine-shift ranges from 35 to 45 tons, depending on the nature of the ground.

Squibbing and Shooting—In drilling in the flint of the sheet ground the steel frequently encounters shelly ground or fissures. This causes the steel to stick and, if the trouble cannot be remedied by re-aligning the machine, the steel is removed and a stick or half stick of powder is exploded in the hole. This practice, called squibbing, usually removes the obstruction. Owing to the toughness of the rock it is necessary to chamber the holes in order to introduce sufficient powder into them to break the ground. The chambering of holes is also referred to as squibbing.

While the practice varies, it is found to be most satisfac-



SKETCH 2. ENLARGED PLAN OF WORKING FACE

tory to drill a round of holes, squib them when the shift goes off, and shoot them on the following day after the shift has left the mine. Thus a round of holes is carried a day ahead of the final blasting. Where two shifts are worked, the round is usually carried one shift ahead of the final blasting. Where squibbing and shooting is done while the shift is in the ground, the men are subjected to great quantities of rock dust.

Carrying Stope and Heading—Deposits from 16 or 18 to 30 feet thick are usually mined with a stope (bench) and heading. Sometimes there may be two benches and a heading. While local conditions require slight variation in practice, the method used in one large mine will serve to typify the mining of thick deposits.

The thickness of the deposit is 24 feet (see Sketch 3, Fig.

B). The rock, the usual cherty flint of the sheet ground area, is uniformly hard with the exception of a small band of what is called cotton rock, which appears in the floor of the heading. The roof, also of flint, is fairly good and from 3 to 4 feet thick, limestone lying above. The heading is carried 8 feet high, the drill holes being 8 feet in length. This heading is carried from 15 to 20 feet in advance of the stope. The practice here is practically the same as that already described for mining in the heading.

The lower bench, called the stope, 16 feet thick, is broken with stope and splitter holes about 18 feet long. The splitter is pointed slightly upward; the stope hole slightly downward. These holes are squibbed three or four times, first with 4 to 5 sticks of powder, increasing to about 30 sticks for the last squibbing. Four of these 18-foot holes, two splitters and two stope, loaded with from three to four boxes of powder each, will break the 16-foot bench across a width of 10 to 12 feet, producing from 35 to 45 tons of dirt. The time required to drill one of these 18-foot holes is from $1\frac{1}{2}$ to 8 hours, depending on the nature of the ground. In mines where the bench is not so thick as here, say 10 feet, the splitter hole is not used, one stope hole being sufficient to break the ground.

A different method of attack, not usually practiced in the district, was noted in this mine. The change of method, which resulted in much cheaper dirt, was necessitated by a change in the nature of the rock. The upper 12 feet of the deposit became extremely tough, while the lower 12 feet could be broken easily. It was decided to carry the heading in the softer ground, and the stope in the tougher ground above. A section of the face is shown in Sketch 3, Fig. C. The heading is carried in the usual manner, holes being drilled 12 feet long. The upper strata is broken with roof and splitter holes from 12 to 14 feet long. The splitters are squibbed, but the roof holes are left as drilled, as squibbing would tend to loosen the roof and make it dangerous. The heading is carried from 50 to 100 feet ahead of the stope. This method of mining produces the cheapest dirt in the mine. Five machines produce an average of 650 (1,000-lb.) buckets of dirt in 8 hours. This amount of dirt is handled by 11 shovelers.

Explosives—In most of the mines two kinds of dynamite are used—ammonia and gelatin of from 33 to 40 per cent. strength. The ammonia dynamite is used for dry, and the

gelatin for wet holes. In exceptional cases higher strength dynamites are used, especially in long stope holes. In some of the mines 80 per cent. gelatin dynamite is used for squibbing.

In the sheet ground mines explosives constitute one of the chief items of mining expense, varying from 20 per cent. to

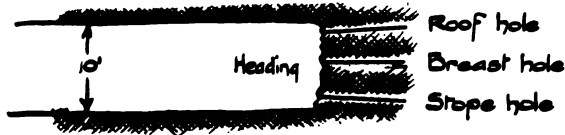


Fig. a

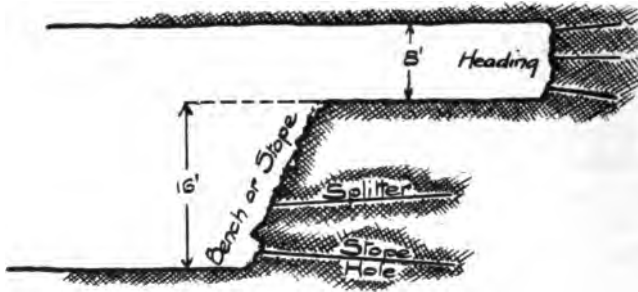


Fig. b

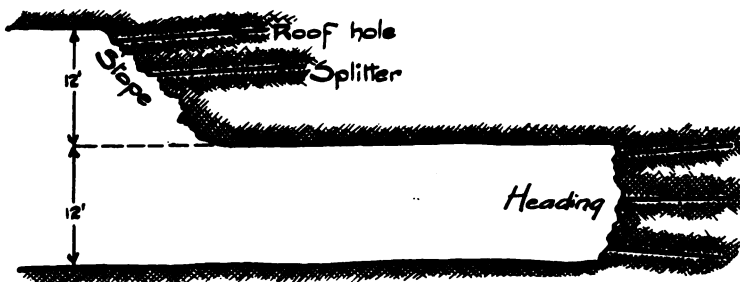


Fig. c

SKETCH 3. METHODS OF CARRYING FACE

as high as 30 per cent. of the total cost. A box of powder breaks from 30 to 45 tons of dirt, according to the nature of the ground and the kind of face carried.

Pillars—In determining the size and number of pillars to be left in the mines, the chief factors to be considered are the thickness of the deposit and the character of the rock. Usually there is no set rule, pillars being spotted to meet conditions as they develop. The tendency in recent years has been to leave pillars from 40 to 60 feet apart, arranged in "five spot" form, as shown in Sketch 2. They may be from 15 to 20 feet, or more, in diameter. Formerly pillars were placed at much greater distances apart (in some cases as far as 100 feet) and in well defined rows. In Sketch 2 the pillars are marked "P"; at the points a, b, c, d and e, pillars are being blocked out. From 10 to 30 per cent. of the ground is left in pillars. It is customary in some of the mines to allow leasers to make a clean-up of floors and pillars after the company work has ceased. Frequently leasers will cut a pillar down to a width that allows for little or no factor of safety. There have been some serious caves as a result of this practice. Were it not for the vigilance of the mine inspectors, doubtless there would be more.

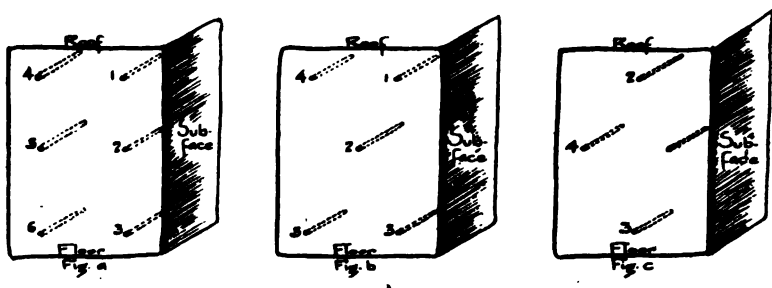
The spotting of shaft pillars is not always carried out with a view to future convenience of operation. In some mines the richness of the ground around the shaft station has been the cause of the removal of rock that should have been left in pillars. In exceptional cases shafts have been left almost entirely unprotected by pillars. While the best practice varies according to conditions, the two arrangements of shaft pillars shown in Sketch 1 appear to be satisfactory for ordinary needs. The two pillars shown for the shaft at the left would be about 20 feet thick and 30 feet long for a roof height of from 7 to 10 feet; in thicker deposits these dimensions might be doubled. The use of four shaft pillars, as shown at the right, allows for haulage in any direction.

SHOVELING, TRAMMING AND HOISTING.

The broken rock, or dirt, is shoveled to buckets of from 1,000 to 1,650 lbs. capacity. Most of the dirt is small enough for handling with the shovel. Boulders that cannot be broken with a sledge are broken with powder, this practice being called boulder popping. The shovelers, or "cokeys," as they are called, are paid by the bucket, and load all the way from

30 to 70 or 80 buckets in 8 hours. There are some records of 90 and 100 buckets per shift. The average for all of the sheet ground mines is 22 tons per 8 hours per shoveler.

Usually the shoveler trams his buckets, on small trucks, to a switch, or lay-by. Here the loaded trucks are collected and hauled to the shaft station by mules. In smaller mines



SKETCH 4. PLACING DRILL HOLES

the shoveler trams his bucket to the shaft. In one mine a gasoline motor collects and hauls the buckets to the shaft; in another rope haulage is used.

Tracks are from 12 to 24 inch gage, 8 to 12-lb. rails being chiefly used. There are two methods of spotting the loaded truck at the shaft station. One is to arrange for stopping the truck at a point in the exact center of the shaft, so that in hoisting it is not necessary to steady or balance the bucket after it leaves the truck. The other method is to land the empty bucket, as it comes down the shaft, in the center, and hook on to the loaded bucket just beside it. This makes it necessary to stop the loaded bucket momentarily a few feet from the truck and steady it in the center of the shaft.

Derricks (headframes) range from 40 to 70 feet in height, depending on the elevation at which the dirt must be delivered to the mill. Geared steam and electrically operated hoists are chiefly used, although there are a number of first motion hoists in use. The hoisting engine is placed in the top of the derrick, so that the engineer at the lever, or controller, has an unobstructed view to the bottom of the shaft. Steel cables, from $\frac{1}{2}$ to $\frac{5}{8}$ inch in diameter, are used.

The operation of changing the cable hook from the empty

to the loaded bucket is very quickly done. Dirt is hoisted without signals, the engineer apparently knowing just how much time to allow before hoisting. There is practically no lost motion in the hoisting. It is common practice to raise and dump 120 buckets and more in one hour. In handling men in the bucket, the engineer is able to keep his eye on his load. If the bucket gets too close to the side of the shaft the speed of hoisting is reduced to give the men a chance to center the bucket.

PUMPING.

Various types of pumps are employed in the mines: these include the Cornish type, electrically-driven centrifugal, and steam plunger pumps. There are one or two installations of plunger pumps driven by gas engine; these are placed underground in well ventilated parts of the mine. The mines make upward of 1,500 gallons of water per minute, this being a maximum figure. In localities where several mines are cut together, pumping is done from a central station.

SANITATION AND HEALTH.

Conditions in the Mines—With the exception of the prevalence of siliceous rock dust in the mine air, the sheet ground miners work under the most favorable conditions. Although trouble is experienced in some places on account of powder gas and smoke, the mines are generally well ventilated. Temperatures the year round range from 55 to 65 degrees, Fahrenheit, relative humidity varying from 85 to 100 per cent. The humidity is usually high at the working face, especially where the air current is sluggish, but this is not serious owing to the low temperatures.

It is seldom that a miner finds it necessary to work in dripping water. The average shoveler will work in a puddle of water if there is one handy to his pile of dirt. The water seems to cause the dirt to run to the shovel more readily.

Surface Conditions—Change-houses are generally small and lack conveniences for washing and for the drying of clothes; in some cases they are kept in an unclean condition. These faults, however, are being remedied at many of the properties. Quite recently some model change-houses have been erected.

The living conditions of the miners and their families are not what they should be. The unmarried men usually live in boarding houses; those with families rent, and in some cases

own, their homes. The miners are all Americans, recruited from Missouri and nearby states. There are a few from western mining districts.

Rock Dust and Pulmonary Diseases—There is probably no mining district in the United States where pulmonary diseases are more prevalent than in the Joplin district. It is variously estimated that from 30 to 50 per cent. of the miners are affected. This condition was brought to the attention of the U. S. Bureau of Mines, and late in 1914 an investigation was started by this bureau in co-operation with the U. S. Public Health Service. The latter bureau assigned to the work Passed Asst. Surgeon A. J. Lanza; the writer represented the Bureau of Mines.

This investigation developed into an educational campaign which has now extended over a period of nine months. The chief fact brought out through the investigative work was that the siliceous rock dust in the mines is the prime factor in causing miners consumption and tuberculosis among the miners. This rock dust was produced chiefly by drilling, shoveling, squibbing, blasting, and the blowing of drill holes without water.

Thanks to the active co-operation of the State mine inspectors, the operators, the miners, and various organizations, conditions in the mines have been very much improved. A sanitary and safety organization has been perfected among the operators. Through illustrated lectures, and talks to miners at change-houses, the men have been impressed with the seriousness of the situation and the necessity of keeping down the rock dust. Dr. Lanza has established, and is now maintaining, a clinic at which miners are examined and advised free of charge. State laws have been passed requiring that a separate water line be carried to every working face in dusty mines. This law provides a penalty for the operator who will not install the water lines, and a penalty for the miner who will not use the water when it is furnished. The blowing of dry holes is made a misdemeanor. Another law provides for change-houses of adequate size and proper equipment. These laws went into effect July 1, 1915. Practically all of the dusty sheet ground mines are equipped, or are preparing to put in equipment, for the abatement of rock dust.

WAGES AND COSTS.

From the beginning of 1912 to early in 1915, drillmen re-

ceived from \$2.25 to \$3 for an 8-hour day; drill helpers were paid from \$2 to \$2.50 per day; and shovelers from 6 to 9 cents per bucket. During this period the zinc concentrate produced was sold at an approximate average of \$43 per ton; lead at \$52 per ton.

During this same period the cost of mining and milling a ton of dirt ranged from 80 cents to \$1.25. A fair average for the larger operations would be \$1 per ton. Thus, 100 tons of dirt would cost \$100 for mining and milling. If this were 2 per cent dirt it would yield 2 tons of zinc concentrates which, at \$43 per ton, would total only \$86. Obviously, at this price of ore, the cost of mining and milling must be kept below the average if there is to be any profit made on 2 per cent. dirt.

In the first quarter of 1915 the price of zinc ore (concentrates) began to climb upward, owing to the great demand for spelter occasioned by the European war. Sales have been made up to this writing (July, 1915) as high as \$135 per ton. The price of labor also has increased. Drillmen have been paid \$3.50 to \$4.50; drill helpers, \$2.75 to \$3.75; shovelers have received as high as 15 cents per bucket.

The following approximate figures show the average mining cost per ton of rock won by one company during 1912, 1913 and 1914: Ground boss, 1 cent; drilling, 18.5; blasting, 16; roof protection, 0.7; shoveling, 15; mule haulage, 6; track-work, 2.3; hoisting, 4; lighting, 0.65; sundry, 1; total mining cost, 65.15 cents per ton. Milling averaged 27 cents per ton, making the total for mining and milling 92.15 cents.

A summary of the cost figures of one large company for a number of years shows the cost of mining divided as follows: Labor, 33 cents per ton; explosives 10c; air, 4c; other expenses, 8c; total 55c per ton. Milling averaged 30c. The cost of mining and milling, including everything except depreciation, was 98 cents per ton.

Still another record, for the year 1914, is as follows: Mining and milling per ton of rock: Labor, 49.15 cents; explosives, 20.49; fuse, 0.57; gas and electricity, 14.28; superintendence and repairs, 12.12; other expense, 3.39; total, \$1.

The cost of mining and milling, with labor at the high figures that prevailed in the first half of 1915, increased to \$1.50 upward to \$2 per ton, and higher at some properties.

COMMENTS.

A mine operator from a district where extensive equipment is used, on going into a mine in the Joplin district, is impressed with what he believes to be a lack of proper equipment, and the corresponding crudeness in the methods of handling rock. After a glance at the cost sheets, however, this operator will doubtless feel a desire to look again at the mine equipment and the methods of operation. As a matter of fact, extensive mine equipment does not pay in this district, and it would be a very difficult matter to improve greatly on the methods in use—as far as costs are concerned. The ore deposits are comparatively shallow and can be worked to better advantage through a number of shafts simply equipped. There are few metal mining districts in the United States where ore can be mined and milled for \$1 and less per ton, or where the output averages 10 tons per man employed underground—and this is true for the sheet ground mines of the Joplin district.

The writer is indebted to the State mine inspectors and many of the operators of the Joplin district for much of the information contained in this paper, and takes this opportunity for expressing his appreciation and thanks for the generous treatment accorded him.

DISCUSSION.

MR. KELLY: I would like to ask whether these mines are handled by a single shaft or whether more shafts are opened?

MR. HIGGINS: The custom is to use many shafts and to equip them very simply. It has been demonstrated that expensive equipment will not pay in that district. The average distance apart of shafts is about 300 feet.

MR. KELLY: They are connected through?

MR. HIGGINS: Yes, and the ventilation is good.

THE OPENING OF THE WAKEFIELD MINE.

BY W. C. HART, WAKEFIELD, MICH.*

As open pit mining on an extensive scale has been unknown heretofore on the Gogebic Range, mining men generally have been interested in the recently developed Wakefield mine, at Wakefield, Michigan. This property is in Section 17 of Township 47 North, Range 45 West, Gogebic County, Michigan. Lying, as it does, a half-mile south of the formerly supposed line of the footwall in this portion of the range in a district where mining has been going on for 30 years, the new property does credit to the knowledge and persistence of Mr. Robert Selden Rose, geologist, of Marquette, through whose instrumentality and under whose direction the drilling, which developed this orebody, was undertaken by the men now forming the operating company.

The history of earlier attempts to locate the footwall and find an orebody in this south area would be interesting if space permitted. Such attempts have been made at various times for a number of years. The Pinton Brothers put down some drill holes in Sec. 16 west of the village of Wakefield with the idea that ore existed south of Sunday Lake. They even sunk a small shaft near the greenstone outcrop south of the Chicago and Northwestern depot at Wakefield, going through some iron formation that looked promising. The shaft and drilling were abandoned before definite results were accomplished.

Exploration was also attempted on Sec. 17, principally by test-pitting. The nature of the surface material made it impossible to sink these pits to any great depth and the work was abandoned before ledge was struck. Several of these pits were still in evidence when the writer first came to the property. Some diamond drilling was also done on Sec. 17, the hole which came nearest to finding the ore, being an inclined one pointed toward the present footwall, 100 ft. west and

* Superintendent, The Wakefield Iron Co., Wakefield, Mich.

about 800 ft. north of the present "A" Shaft and cutting under the orebody now developed.

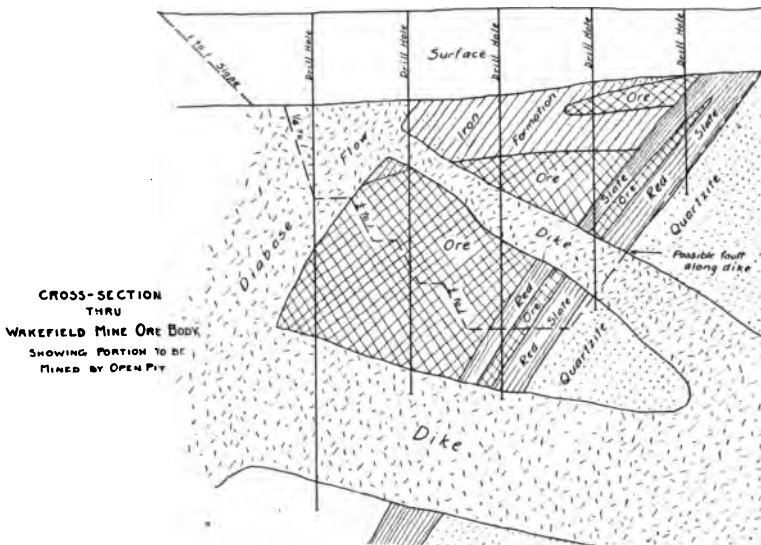
The first drill hole of the successful exploration was located 500 ft. due south of the hole put down by the last exploration company. Drilling was started in this hole on July 16, 1912. A Keystone churn drill was used for the work, and the results obtained were sufficiently satisfactory to warrant continuing the drilling of the property, first with one Keystone drill, and later with several diamond drills; drilling being carried on until sufficient ore had been developed to warrant taking out a lease and sinking a shaft.

The first shaft was started on February 19th, 1913, about 300 ft. south and 100 ft. east of the first drill hole. There are no especially interesting features in connection with the sinking of this shaft. It is a rectangular, vertical, timbered shaft 12 ft. by 18 ft. outside dimensions consisting of 5 compartments, namely two for the hoist, one for a cage, one for ladders and one for pipes and wires. The first 30 ft. of the sinking presented some difficulty on account of the quicksand, but thereafter the progress was unbroken and stations were cut and levels opened at depths of 80 ft., 150 ft., and 250 feet. Fifteen thousand tons of ore were mined on the 80-ft. level for the first shipment from the property in October, 1913.

On August 1, 1913, a second shaft was started 2,000 ft. east of the first. As there were 90 ft. of surface at this point, a large percentage of which was quicksand, a drop shaft was determined upon. This shaft was solid-timbered and rectangular with a bell-mouthed, steel-shod, sinking shoe. The dirt was hoisted by a bucket swung from an aerial tram. Some of the features of this sinking might be of interest, but as the purpose of this article is to deal primarily with the open pit, a detailed description of this operation will be omitted. The shaft was finally ledged, at which point the size was increased to 12 by 18 feet. Sinking was continued to the 400-ft. level with steel sets, and exploratory work begun. Up to the present date, no ore has been shipped from this shaft, although some exploration work has been done by rock drifting.

While the underground development had been going on at "A" Shaft, drilling on the west half of Sec. 17 had shown the possibility of an orebody capable of development by open pit methods. This necessitated the extension of the plans

of drilling for several reasons. First, it was necessary to determine the exact nature and extent of the orebody in order to figure the quantities and the nature of the overburden; second, the nature and extent of the intrusive dikes must be determined in order to be certain that these dikes would not interfere with the removal of the pit ore by steam shovel methods; third, it was necessary to know the quantity of the ore tributary to steam shovel operation in order to plan the limits of the stripping and to determine whether the proposition was practical from an operating and from a financial standpoint. Close drilling was also necessary in order to



lay out the plans of the pit in such a manner that merchantable ore would be available at all times during the life of the pit and a continuous operation assured. This drilling would have added an unnecessarily heavy investment charge to the property had underground methods still been found necessary. It was, however, absolutely essential to the proper development of an open pit mine in a district where the problems likely to be encountered were practically unknown.

When exploratory work had progressed sufficiently to warrant a certain amount of stripping, a contract was let to the Butler Brothers, who are among the oldest stripping contractors on the Mesabi Range, and stripping was started on Oc-

tober 1, 1913. Butler Brothers' equipment for this work consisted of two 100-ton Bucyrus steam shovels, four 50-ton Baldwin locomotives, Western Wheeled Scraper Co. 20-cubic-yard air-dump cars and 85-lb. steel rail. The stripping operation was pushed continuously throughout the winter months, and the first ore uncovered on April 7th, 1914. By July 10, 1914, sufficient ore was uncovered to enable The Wakefield Iron Company to start one of its own shovels in the ore. The ore uncovered at that time was small, as the stripping operators had not sufficient time to clean up a very large area. For this reason the ore loading was slow and uncertain, but operations were carried on, more or less continuously, until the close of navigation, the first year's shipment from the pit being 240,000 tons.

During the shipping season and until January 1, 1915, Butler Brothers continued the stripping operation, completing a contract for approximately one and a half million cubic yards of overburden. On January 1, 1915, at the completion of the Butler Brothers contract, The Wakefield Iron Company took over the stripping work and put two Model 85C Bucyrus shovels and four Baldwin 50-ton locomotives at work on double shift to clean up sufficient ore for the 1915 shipment. A third shovel had also been purchased for the ore operation of identical type to the two engaged in stripping. A number of Western Wheeled Scraper Company's 20-cubic-yard air-dump cars and several hundred tons of new 85-lb. steel rail, in addition to all the steel purchased from Butler Brothers, completed the stripping equipment. Stripping was pushed continuously throughout the winter months and an additional 500,000 cubic-yards of overburden removed before the opening of navigation in 1915.

At the opening of navigation one shovel was left on double shift on the stripping operation and two shovels with one crew on each shift were started in the ore. This enabled the pit to produce the mixture of ore required for sale contracts without unduly moving the shovels by alternating one crew between the two. The use of two shovels in the ore also made it possible, in case heavy shipments were required for short periods of time, to put two crews in ore until the requirements were filled. Later in the season a fourth shovel was added to the equipment. This is a lighter model, a 70 Bucyrus, to be used mostly in clean-up cuts on the ore and in the lighter stripping cuts. With two shovels in ore and

two in stripping a steady operation can be maintained, even when ore shipments become slack and there is nothing for the ore shovels to do, without the expense of moving a shovel, the entire crew can go on to the stripping operation until more ore is required. The fact that the company is carrying on both ore and stripping operations makes for great flexibility in the work and tends to reduce the cost of both. This season, to August 20, 1915, about 275,000 tons have been mined from the open pit.

The body of ore tributary to the open pit, as well as the underground ore at the Wakefield mine, is similar in its formation to other orebodies on the Gogebic range. A quartzite footwall dipping about 60 degrees to the north, striking east and west and overlain by a band of red slate, varying in thickness from 10 to 40 ft., is intercepted by a dike striking parallel to the footwall and dipping about 25 degrees to the south. The ore concentration has taken place in the trough formed by this footwall and dike. The whole north side of the open pit orebody is covered by a flow of diabase, forming the hanging or capping.

For the early development of the pit an area was chosen which represented a minimum amount of stripping and a maximum amount of clean ore, with few intrusive dikes and little overlying rock and lean ore. The orebody was opened under the best obtainable conditions, and an opportunity afforded to study the detrimental features as the pit widened and deepened into an area not so favorable to pit operation. The shallowest overburden was found just west of the present "A" Shaft, at the contact of the ore with the footwall. At this point there is only 40 ft.; westerly the overburden increased in depth on the footwall side, reaching a maximum of 115 feet. The top of the ore drops gradually to the north from the contact with the footwall, until it reaches the diabase flow. Here it dips sharply under this flow, so that the north limit of possible stripping is arbitrarily determined. The ultimate plans contemplate a maximum depth of stripping on the north side of the pit of from 150 to 175 ft., this depth taking the cut well into the diabase flow. There is not sufficient ore under the diabase to warrant stripping any greater depth.

The small amount of ore under this diabase and beyond the stripping limits will be scrambled into the completed pit and picked up by steam shovels as in ordinary stockpile load-

ing; some of the ore remaining in the bottom of the pit after scrambling is completed, will be milled to a level driven under the final bottom of the pit; and the balance of the ore, inaccessible by any of the above will be mined by the usual underground methods.

Unlike the larger pits on the Mesabi range, in which a circular system of tracks is possible, depth in the Wakefield pit will have to be gained in the haulage system by the use of switch-backs. The pit being long, narrow, and extremely deep, there will have to be several of these switch-backs to reach the bottom. These will tie up some of the ore which would otherwise be tributary to the open pit steam shovel tonnage. This will have to be mined by hand from the slopes, and milled after the shovel operation is completed. The track system for the final layout contemplates the use of a maximum gradient of 3 per cent.

The problem of handling surface drainage, so as to minimize the quantity of water flowing into the pit, as well as to care for such water as flows from the banks and within the limits of the pit during spring freshets and in times of heavy rainfall, is one to which all open pit mining is subject, and is especially important in the Wakefield pit. Situated in the bottom of a natural drainage area, the pit was subject to a flow of several million gallons of water within its limits for every inch of rainfall. This added to the difficulty of the operation and was also detrimental to the ore, owing to its high porosity and its capacity to absorb moisture. A definite relation was found between the amount of rainfall and the moisture in the ore.

To drain the ore, as well as take care of the flow of rain water through the pit, a system of drifts was developed on the various levels. At the lowest point in the pit a Keystone drill hole, cased with 5 $\frac{5}{8}$ -in. pipe, was put down to the 250-ft. level. A valve was put on the casing and a sump built. This was connected to the main sump at the shaft by a system of launders. The casing was also cut on the 150-ft. level, a valve being put on the pipe at this point, and a similar arrangement of sump and launders built to carry the water to the auxiliary pump station at this level.

In times of normal flow, all the water is carried direct to the 250-ft. level and pumped to the drainage ditch. In times of rain the valve on the 250-ft. level is partly closed to allow only such water to flow to this level as can be handled by

the present pumps. The balance of the water is backed up to the 150-ft. level pumps. In case of extraordinary rains, where the pumps on both levels are unable to handle all the water, the valve on the 150-ft. level is partly closed allowing the surplus water to accumulate in the bottom of the pit until the extreme flow of water abates. The pumps having relieved the water from the pit the valves on both levels are opened, and the normal system is resumed.

As a further precaution the pump station on the 250-ft. level is 7 ft. above the main drift, and a horse-shoe shaped sump drift, 6 ft. below the main one, carries the incoming water around the shaft, from the shaft station to a point under the pump station. In cases of flood this would allow the whole bottom level to serve as a reservoir without drowning the 250 ft. level pumps. No originality is claimed for this arrangement, as it is universally adopted in good mining practice where a heavy flow of water is anticipated.

To minimize the amount of surface water flowing into the pit, a steam shovel draining ditch was started in the low country 1,000 ft. east of the pit and dug on an average grade of 5 per cent. west along the south final limits of the pit. The pit itself lay in so deep a hollow that this ditch, in order to maintain its grade, finished at a point several hundred feet south of the final limits. To prevent the water in the unprotected area between the ditch and the pit from flowing into it, a waste dump of stripping material was started on the north side of the ditch and carried north on an ascending grade, so that the water falling upon it flows to the ditch. It empties into the Little Black river at the extreme east end of the property.

As a final precaution, a berm is planned on the south side of the pit with a uniform grade the entire length, starting at the west and ending at the east end. In this berm a launder will be built of sufficient size to carry all the water not caught by the steam shovel drainage ditch.

The red slate overlying the footwall quartzite, and the intrusive dikes throughout the orebody, are two factors not encountered in a Mesabi pit. The red slates, as previously mentioned, lie close on the footwall and vary in thickness from 10 to 40 ft. All of this material must be moved in the development of the orebody. The footwall, especially at the western end of the pit is steep; and this slate disintegrating and slipping into the ore, contaminates it and retards the op-

eration by the vast amount of hand labor necessary to separate it from the ore in case it is not removed on each successive bench. Where the line of demarcation between this slate and the overlying ore is not well defined, hand labor is necessary to make a clean separation. The same is also true in the case of the intrusive dikes. In planning track grades for the steam shovel cuts, careful determination of the position of these dikes must be made in order to make separation with a shovel possible and to avoid excessive hand labor. Some portions of these dikes are ferruginous, running as high as 40 per cent. in iron. Such material is put on the No. 2 lean ore stockpile for use in time to come when it will have a commercial value. Other portions of the dike, containing no iron, are sent to the waste dump.

The diabase dike, overlying the north slope of the ore-body, is soft at the top, growing harder as the cuts deepen and finally reaching the hardness of solid ledge at depth. The cost of removal of this dike is great as compared with ordinary stripping material and care had to be exercised in determining the north stripping limits to avoid a total cost which would be excessive for the amount of tributary ore. The separation of this dike from the orebody causes no particular difficulty, as the line of demarcation is well-defined.

The grading of the ore in an open pit on the Gogebic range is more complicated than in most pits on the Mesabi range. The only analysis of ore in each successive cut is that obtained by a study of the nearest preceding one, the nearest drill hole sections, the results of such auxiliary pits as have been sunk in the area, and the underground drifts nearest the cuts. The ore in several cars loaded in one move of a shovel often varies as high as 3 per cent. in iron and as much in manganese in occasional high manganese areas. Unlike an underground operation, where only such places are mined as will produce the desired grade, the shovel must push through the cut regardless of the ore encountered, in order to develop the proper grades for tracks and to shape the pit properly. If the ore is undesirable for immediate shipment it must be stocked on surface and picked up later by a steam shovel, at times when direct pit ore is of such grade as to warrant the mixture. The above conditions necessitate a thorough grading-system both at the mine and at the docks, and involve numerous clerical details in keeping account of the analyses of all cars leaving the mine and their position in the

dock so as to load the boats with only the desired ores and to get a uniform mixture. Considering the conditions, lower lake checks on cargo analyses under this grading system have been very satisfactory.

As to the ultimate possibilities of production of ore by steam shovel methods from the Wakefield pit, plans have been made so far as they are possible with present information. These are based on the removal of overburden to a depth of from 150 to 175 ft. on the north side of the pit, as previously stated. Slopes are figured in surface material at 1 to 1; in hard dike and rock at $\frac{1}{4}$ to 1; and in ore at $\frac{1}{2}$ to 1. As a rule 25 ft. berms are used for the protection of the pit, except in cases where track benches form the berm. Track grades are planned on a maximum of 3 per cent. The depth to which ore can be mined and consequently the tonnage tributary to open pit methods, will depend on the above factors.

If the angle of repose of the various materials conforms to the above figures, the estimated tonnage will hold out, except as that tonnage may be cut down by intrusive dikes at present unknown. If these slopes do not hold out, a less tonnage will be available for open pit operation and a greater tonnage will have to be mined by milling and underground methods. This, and a number of other important factors in the development of the Wakefield pit, will depend on the conditions which will arise in the future. The pit is in its early stages and all the problems connected with its development have by no means been encountered nor solved.

DISCUSSION.

MR. DENTON: I haven't read the paper, but you speak about the equation between the ore and the stripping. Did you enter into that in the paper?

MR. HART: No, I did not. It is a detail that ordinarily isn't published, although I did not omit it for that reason. I omitted it because it is a detail that it did not occur to me to mention.

MR. DENTON: I was wondering whether you had determined the profitable depth of stripping?

MR. HART: Not exactly, but the limits will be extended to such a point as will make a good proposition as an open pit operation.

MR. DENTON: For the extent of stripping already done, have you estimated the amount of ore uncovered?

MR. HART: No, we have not. In a pit of this size we are so far ahead with our stripping that the amount of ore actually tributary to the present stripping done is relatively small owing to the great amount of ore tied up under the slopes and the stripping benches. In other words, the first two million yards would uncover a comparatively small amount of ore. I figured roughly when we started to strip, that from the start of our approach to the west end of the property, figuring the normal angle of the material, if we excavated a triangular piece of ground coming to a point at the ore, we would have very close to half a million yards. There is half a million yards tied up in slopes at the smallest depth. With our material, the way the ore has stood up on the north bank, that bank has taken a one and one-half to one slope in a good many instances. We have stripped far enough back to enable us to carry all of the tracks necessary. In our proposition, we cannot crowd the banks with our ore operation, as is done on the Mesabi. Our depth of surface increases so fast as we extend to the north, that as many as four stripping benches will have to be maintained until the final slope is reached on the north side.

MR. DENTON: You haven't yet fixed a standard as to the amount of stripping to the unit of ore?

MR. HART: Only the standard of dollars and cents.

MR. DENTON: You haven't established a ratio between ore and stripping?

MR. HART: That will change from year to year. If we found that the amount of diabase was so great, or the quality of that diabase became so hard that it was costing us a dollar a yard to strip it, we would have to change our plans. The amount of stripping that can be done for a certain quantity of ore is a question purely of the cost per yard of the stripping. The old quotation of "a foot of stripping for a foot of ore" is obsolete, and the only thing that can decide the ratio is the amount of stripping cost which must be charged to the ton of tributary ore.

MR. DENTON: To put it another way, how much do you estimate you save in the cost of mining by stripping?

MR. HART: If we carried it to an extreme, we would save only in the difference in labor and timber, and in interest on the investment. Suppose we carried our stripping to a point where it equalled the cost of underground operations, so far as cost per ton in actual producing cost was concerned,

we would still gain by the fact that we would require less labor; we would be able to produce ore quicker and we would make a greater volume of ore available for shipment at any time.

MR. DENTON: Did you go at all into the question of change in slopes of your banks for different depths?

MR. HART: We decided on an ultimate plan that would include all of the stripping that is to be done if conditions were ideal, but we will have to modify that plan as conditions arise which militate against the ideal development of the pit.

MR. DENTON: I was wondering whether you had gotten to the fine point of figuring different slopes at different depths, even if the material is the same.

MR. HART: We have figured on that. The slopes will be different. We have about 20 ft. on one side of the pit which is pure sand, which will undoubtedly take a slope of one and one-half to one. Below that we have clay that will probably stand on a slope of three-quarters to one. We have taken that into account, although the average of the bank down to the diabase, we figure at a slope of one to one.

MR. DENTON: Do you protect your berms?

MR. HART: On the footwall side, but not on the hanging-side. We had no trouble at all there. Eventually on the footwall side, as we go down, we will undoubtedly have to rip-rap the full length of the pit. I have in mind the Buffalo & Susquehanna on the Mesabi, where they have a stone hedge surrounding the pit on the berm to prevent material from dropping over in the deepest part. We will undoubtedly have to do something of that kind. We have done it to some extent in the development of the pit with lagging and blocking.

MR. VANEVERA: Approximately how long and how wide is the pit now?

MR. HART: The pit is approximately 2,100 ft. from the shaft to the west end of the property. It is 600 ft wide at the extreme width.

MR. VANEVERA: On the ore?

MR. HART: No, on the top of the stripping. It is approximately 300 ft. at the widest point on the ore before the ore dips sharply to the north.

THE USE OF GUNITE IN A STEEL SHAFT AND IN AN UNDERGROUND PUMP HOUSE ON THE GOGEBIC RANGE.

BY STEPHEN ROYCE, HURLEY, WIS.*

The application of Gunitite, or "Gun-crete" as it was formerly called, to mining operations is comparatively recent. Gunitite may be defined as a mixture of sand, cement, and water blown on a solid surface by a high pressure of compressed air forcing it through a hose and nozzle. An essential part of the gunitite coating is the use of some form of reinforcing wire to be applied to the surface to be coated before the cement is blown on. The process was originally invented as a cheaper and more efficient method of applying stucco to a building, but has proven itself adaptable to a great variety of purposes.

The places in which gunitite has been used on the Gogebic Range are the Cary "A" shaft, at Hurley, Wisconsin, and the 18th level pump-house of the Sunday Lake Mine, Wakefield, Michigan.

Fig. 1 shows a cross section of "A" shaft, and a longitudinal section, as well as details of the method adopted in applying the cement coating and its reinforcing. "A" shaft is a steel five-compartment shaft, sunk to a depth of 1,320 ft. in the quartzite. The steel sets are blocked in place with wooden blocking, and the lining of the shaft consists of 3-in. tamarack plank wedged into the flanges of the I-beams, which form the wall plates and the end pieces.

The purpose of the gunitite coating in "A" shaft was, first, to fire-proof the lathing and wooden blocking; second, to protect the lathing from contact with the air in the shaft, so retarding its decay; third, to form an air and water-proof coating over the shaft lining, keeping air from entering the space between the timbers and the rock, and keeping in the water which is flowing along in the same space. This last

*General Engineer, Pickands, Mather & Co., Gogebic Range

feature is expected to retard the decay of the blocking as well as the lathing as this will be water-logged all the time. The gunite coating, by reason of its wire reinforcement, which will be further described, is expected to reinforce the lathing and partially take its place in the event of decay actually occurring. The coating was applied not only to the walls of the

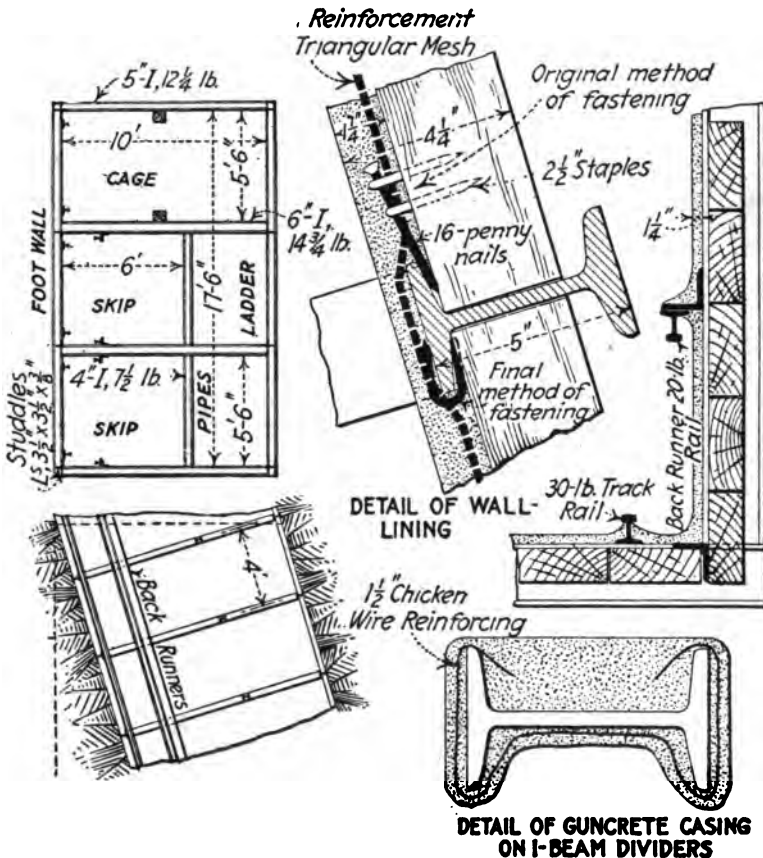


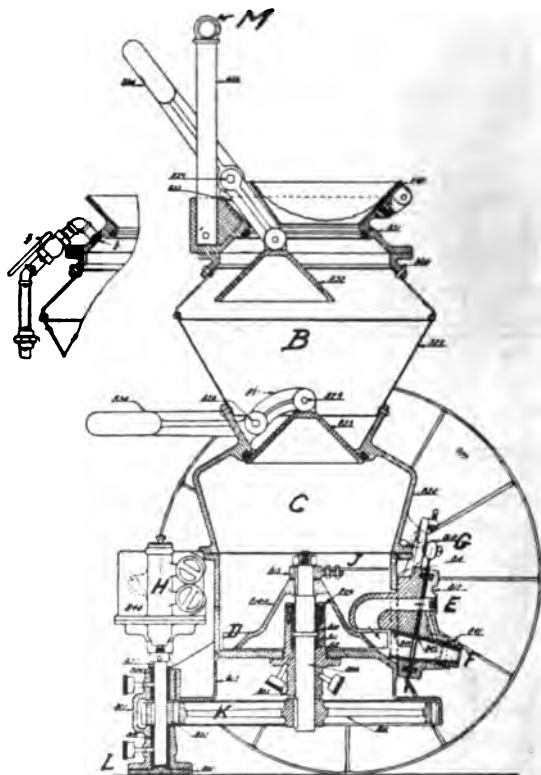
FIGURE 1. SHAFT ARRANGEMENT AND DETAIL OF COATED MEMBERS

shaft, but also to the steel dividers, the intention being to protect these from rust and incidentally reinforce them after the manner of a reinforced concrete beam.

At Sunday Lake the purpose was to fire-proof the pump-house, which is timbered with heavy wooden posts, caps and

lagging, and to protect the wood from dry-rot, which at the high temperature in the pump-house, quickly attacks it.

Fig. 2 shows a view of the apparatus which is known as a "Cement-gun." The apparatus consists of an upper, or feed-hopper, (B), a lower discharge hopper, (C), and feed-wheel, (D), which is geared to the motor "H." The motor "H" is worked by an air pressure of from 45 to 75 lbs. to

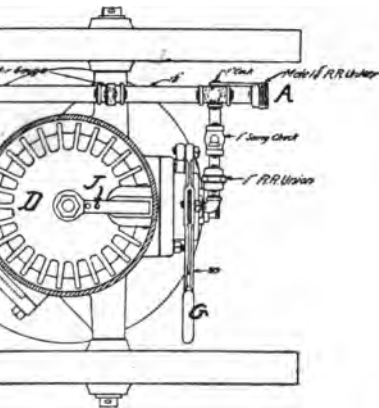
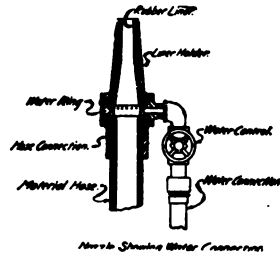


Sectional Elevation

FIGURE 2A

the square inch which is supplied to the machine. The discharge hopper "C" is kept constantly under air pressure. The hopper "B" is put under pressure only after the charge of dry cement and sand mixed in four to one proportion has been introduced at the top and the upper opening has been closed. The cement-sand mixture is charged into the top of the feed-

hopper with the upper cone admission valve open, as shown in the picture, and the lower discharge valve, also conical, closed, as shown there. The upper valve is then closed and air pressure is admitted to the feed-hopper until the pressure in the feed and discharge hoppers is equalized. The weight of the cement and sand in the feed-hopper presses down the lower valve and the material drops through into the discharge hopper



Plan of Base Section

FIGURE 2B

per "C," falling on top of the feed-wheel "D." When all the material in the feed-hopper has gone through, the lower valve is closed by means of the lever which is heavy enough so that it usually closes the lower valve with its own weight. The upper valve of hopper "B" is then opened, after turning the air pressure off the upper hopper, and a new charge is put in. The feed-wheel "D," driven by the motor "H," through the gear, "K," expels the cement and sand by means of the

discharge nozzle "F" into the delivery hose, which goes to the operating nozzle. The feed-wheel consists of a series of paddles which form pockets. As these pass the stream of compressed air, which is constantly being admitted at "E," they take up measured quantities of compressed air, which force the material through the discharge nozzle into the hose. The form of the operating nozzle with water connections is also shown.

Water, it will be seen does not touch the cement and sand mixture until at the point where it leaves the hose. The



FIGURE 3

amount of the water is gauged by the nozzle operator by means of the valve shown. The proportion of cement and sand is gauged by the original mixture, but changes automatically in accordance with the requirements of the work. The material as fed to the machine must be dry and no water must reach it before the water control at the operating nozzle. The sand must be clean, sharp, and of fairly uniform grains, in order to secure the best results, both in the work itself and in the operating of the machine. An advantage claimed for cement applied by this process is that the mix-

ture is automatically enriched at the point of contact between the gunite coating and the surface which is covered. This is due to the tendency of the sand in the mixture to rebound when striking a hard surface, leaving the cement almost neat for the first $\frac{1}{8}$ or $\frac{1}{4}$ in. of the coating. After this the mixture grades off until it approaches about a three to one, or three and one-half to one composition. It is said that the original mixture is almost immaterial provided that the amount of sand used is three and one-half to one or over, as the ex-



APPARATUS SET UP FOR WORK AT COLLAR OF "A" SHAFT

cess sand will rebound, so that the final composition of a one and one-half inch coating will be about three to one in any event.

Another advantage of the method is that as water is not mixed in until within a small fraction of a second of the application of the concrete to the job, all the setting power of the cement is used in the work, there being no partial setting in the cement box before application.

The delivery hose has to be made of pure soft rubber. Its

long or short life is quite a factor in the cost of operation and depends largely upon the character of the sand which is fed to the machine.

The application of the gunite coating in "A" shaft was done in two experimental sections, one being from the collar of the shaft to the third level, the other from the eighth to the tenth level. In the first the machine was placed on surface and the hose lead down through the shaft to the point



VIEW LOOKING UP "A" SHAFT SHOWING COMPLETED GUN-CRETE WORK ABOVE. REINFORCING SET BELOW BUT NOT CONCRETED YET

of application; in the second the machine was placed on the eighth level.

The first step in the application of the coating was to clean thoroughly the entire surface to be covered, which was done partly with water under heavy compressed air pressure, partly by sand blasting and partly by chipping the rust and accumulated coating off the steel. Next the reinforcement was applied. This consisted of No. 7, American Steel & Wire Company's triangular mesh reinforcing wire for the side walls. This was originally intended to be applied as

shown in the upper part of Fig. 1. This consisted of a nail driven into the space between the tamarack lathing and the flange of the I-beam. This nail was then stapled, together with the reinforcing wire, onto the lathing. This method was discarded as it placed too much reliance on the holding power of the staples in the wood, which might later rot. A method was substituted for this which is also shown in Fig. 1. This consists in stapling the reinforcing wire directly to the steel I-beams, and was used for most of the work. The reinforc-



VIEW OF SUNDAY LAKE PUMP-HOUSE LOOKING AWAY FROM SHAFT

ing wire was also stapled to the laths at intervals between the steel sets.

An important point in the application of the reinforcing wire is that it should be separated by a short distance, say one-eighth of an inch, from the surface to be covered. This is so that the cement can get in behind the reinforcing and form a unit with it. This was accomplished by stapling the reinforcing wire on with nails under it. The dividers, before the cement was applied, were covered with $1\frac{1}{2}$ in. mesh

chicken wire, clamped on with wire clamps. The dividers were filled in completely on their upper faces resulting in their reinforcement for bearing a downward load. This increase in strength we figure at nearly 20 per cent. The work progressed downward and it was found best to coat the entire sidewalls before coating the dividers. The thickness of the coating in "A" shaft was $1\frac{1}{2}$ in. and it was found that the operator was able to gauge this thickness with astonishing



VIEW OF SUNDAY LAKE PUMP-HOUSE LOOKING TOWARD SHAFT

accuracy. The cement was applied in from two to three coats.

The Sunday Lake pump-house was coated with a one and one-half inch coating of gunite, over all the posts, lagging and exposed timber. This was applied over a reinforcing, consisting of $1\frac{1}{2}$ in. mesh chicken wire on the posts, and a No. 7 reinforcing wire, triangular mesh, on the lagging in the roof. Two photographs are given herewith showing the results in the Sunday Lake pump-house, also a view looking up "A" shaft from a little above the third level. This shows the application

of the cement in two coats, and shows the reinforcing wire attached to the I-beams.

The results so far observed have been excellent in both places. In "A" shaft we have a hard, fairly smooth, water-proof, coating, which does not crack with the jar of the shaft in hoisting, and which we believe will greatly prolong the life of the shaft at the points where it has been used.

An important point, in connection with the water-proof character of the coating, is the necessity of leaving pipes at every level to drain off the water and relieve hydrostatic pressure.

An interesting feature of our use of the cement coating is the wide difference between the temperatures to which it is subjected. At the collar of "A" shaft the cement is covered with frost and subjected to a temperature considerably below zero. In the Sunday Lake pump-house the temperature at five feet above the floor is 111 degrees, and this must rise greatly near the roof of the pump-house.

The cost of lining "A" shaft as described came to \$9.2978 per linear-foot of shaft. The total area of wall surface covered was 14260.90 square feet, the total area of steel covered, measured along the contact of the cement with the steel was 3749.96 square feet. The material used was as follows: Sand, 102½ cubic yards; cement, 173 barrels; reinforcing, 14260.90 square feet; chicken wire, 3749.96 square feet. Fastening staples and wire were also used.

The work was accomplished by one foreman and six men in thirty-two working days. The total linear feet of shaft covered were 263.13. The Sunday Lake pump-house, which was done under especially hard conditions, was considerably more expensive per square foot. Both jobs were done on contract by the Cement-Gun Construction Company of Chicago. "A" shaft was an easy shaft for application of the cement-gun coating as there was comparatively little water flowing on the surface of the lathing which was to be covered. If the cement can once be applied and can harden, no amount of water will make any difficulty thereafter, but there are considerable difficulties in making the coating stick to a wet surface. The same company has recently succeeded, they say, in coating a wet shaft for a large southern Illinois coal company. They usually divert the water from a wet point in the shaft to the sump below by means of permanent drains. Sometimes bleed-pipes are introduced into wet strata and later sealed after

the concrete lining has hardened. In some cases they have to drill through the wooden lagging and with iron rods clear the mud which sometimes accumulates and so allow the water to drain down to a lower point in the shaft. In some places they put a water-proof felt sheathing between the cement and wood of the shaft lining; the water drains down between the felt and lagging and the outer surface is kept dry for the application of the cement coating. This felt, being water-proof and rot-proof, and being completely encased between the cement and wood, is a perfectly solid part of the shaft lining.

A cement-gun coating is applicable for water-proofing, as in leaky masonry reservoirs, etc., for cheaply reinforcing steel beams which have shown signs of yielding, and for resisting abrasion, as in coal bunker bottoms, etc. For remedying the disintegration of masonry it has been used a good deal. An interesting use is for covering the plate girders of railroad bridges which have been found to wear rapidly away on account of the abrasion of particles thrown out of locomotive exhausts. It is used for covering buildings, for floors, and for a variety of purposes. For the uses that we have had for it, it has certainly proven very satisfactory.

A SURVEY OF THE DEVELOPMENTS AND OPERATIONS IN THE CUYUNA IRON ORE DISTRICT OF MINNESOTA.

BY CARL ZAPFFE*

INTRODUCTION.

Twelve years have elapsed since the first drilling explorations in the Cuyuna Iron Ore District of Minnesota. Activities of various kinds have marked every year since then. Although during the last twelve months the number of drills operating has dropped off materially, due to the general depression prevailing in the iron ore business, development work is making good headway. This time and this occasion seems therefore a splendid opportunity for the taking of stock and for ascertaining what a dozen years have done for the Cuyuna and for observing how the regard of mining men for the Cuyuna may have changed and how prejudice may have been overcome.

At the outset the Cuyuna district was but a prophecy, and its existence was anticipated long before its discovery. The prophecy gradually became a reality as the rapid developments in the great Mesabi district to the north gradually extended that district further and further westward and toward the Cuyuna district. Later the rapid developments of the Mesabi district for a time overshadowed the results of explorations in the Cuyuna district and thus retarded its progress. One must admit that the Cuyuna in its early days offered many disappointments to the Mesabi prospectors.

The Michigan prospectors of that day generally entertained only long-distance views and opinions—although since that time they have one by one made trips through the geologically viewless territory embraced by the Cuyuna district. Cuyuna prospecting was therefore largely conducted and promoted by a new and different group of individuals.

It appears that nature was once more operating as an

* Geologist, Brainerd, Minn.

even, for the Cuyuna District developments were held back at a time when other iron ores were flooding the markets. As a whole the first Cuyuna ores found were of the poorest kind; but then followed a period when better ores were encountered; and now it is a pleasure to make and herald the statement that only lately have the best ores come to light and that certain recent rather desultory drilling has produced some astonishingly good results—results which indicate far greater possibilities for the South Range of the district than perhaps even the most optimistic have ever dared to claim. And, as has often been stated, the district has been even now but merely scratched. Another factor that has always retarded developments is the great length of the district. Nor must one forget the periodic economic disturbances that affect any large non-bessemer district. It is also true that nearly every one of the substantial mining operations showed better ores as the underground work was extended than was anticipated from the drilling. Of course difficulties and disappointments have never been lacking; but these are common to all districts. The newer and better developments have come during periods of less excitement, and though they have called for no more courage and sagacity, they have probably required more method and more resources.

IRON CONTENT OF THE ORES.

A few years after drilling began, I made an estimate of all the ores developed, and found that the average iron content of all material analyzing 50 per cent. or more was just about 53 per cent. Only two properties on the North Range of the district had been drilled up to that time, one of these being what is now the Kennedy mine.

The following five years was a period of great activity on the North Range, and the average iron analysis was gradually increased to a little over 56 per cent. by the finding of deposits with large tonnages of 60 per cent. ore. Developments on the South Range came to a standstill at this time, on the North Range, numerous properties were opened for mining in the latter part of the period.

The third period of development was ushered in by the finding of a minable deposit of hematite ore almost entirely of Bessemer grade. In this deposit some of the analyses for iron ran over 69 per cent. and some of the phosphorus analyses around 0.010 per cent; also, on the North Range a large

tonnage of the ore developed is contaminated by decomposed chert and on two properties washing plants have been installed to remove this material and other deleterious substances that readily wash out, and thus a large tonnage of otherwise waste material is made usable. The results of recent explorations on the South Range are quite in contrast to this. As has always been known, the ores here are never the sandy or so-called wash ores; and at the present time, the South Range ores are found averaging not only higher in iron but lower in silica than ever before. During the last twelve months a large tonnage of brown ore has been developed on the South Range that analyzes over 60 per cent. iron and under 5 per cent. silica. For example, in one instance in one angle hole the first 35 feet of ore averaged 64.52 per cent. iron and 1.43 per cent. silica and the next 35 feet 60.68 per cent. iron and 6.18 silica; thus 70 feet of ore averaged 62.60 per cent. iron and 3.80 per cent. silica. Anyone familiar with all the facts cannot be other than enthusiastic at the turn that developments have taken of late and it is not wild to predict that the South Range ores will soon at least equal those of the North Range in iron content and may be expected to excel them in furnace value because less apt to contain manganese or too much silica.

CONCENTRATING ORES.

The North Range deposits, as already mentioned, are frequently much contaminated by disintegrated white chert. This occurs sometimes in minute bands, sometimes in bands a few inches wide, and sometimes as banded masses in which the bands of ore are so thin as to be scarcely visible. In some cases this material has been loaded with better ores, and as a result the ores shipped have been undesirably high in silica. Much of this chert is so thoroughly disintegrated that it resembles a white flour when accumulated, and will flow when wet. From much of the ore-bearing formation this cherty material can be removed by washing and two small and simple washing plants have already been erected for this purpose. Washing such an ore carrying as low as 45 per cent. iron dried has raised it to 53 per cent. and over. These plants have barely begun to operate, yet they have already produced favorable results and hence promise well for the so-called Cuyuna concentrating ores. As already stated, such ores are unknown on the South Range.

MANGANIFEROUS ORES.

With a very few exceptions North Range ore deposits are accompanied by more or less manganiferous material. In most of the ore deposits there is a very substantial tonnage of it, usually so localized that it can be left in the mine or handled without contaminating the regular iron ore product. In a few of the properties, however, apparently nothing other than manganiferous formation exists. Some of these are being operated and small amounts have been shipped. At one mine the ores have been graded underground into classes depending mainly upon the manganese content; here the combined metallic units of manganese and iron usually amount to 55 per cent. or over. In another instance plans are under way to attempt mechanical grading magnetically at the surface. More than this can not be stated at this time regarding this process.

Some analyses for manganese have exceeded 50 per cent., but thus far very little material has been shipped that has averaged above 30 per cent., and in the long run it would probably be detrimental to a property to attempt to maintain even only a 25 per cent. grade. There is an almost unlimited quantity of manganiferous material ranging from 10 to 15 per cent.

Manganiferous material is unknown on the South Range. Rarely is even a one per cent. analysis encountered, and many deposits will average under one-half of one per cent.

OTHER FEATURES OF THE ORES AND DEPOSITS.

Much alarm has been spread in the past over excessive moisture. It is true that at their opening some of the properties yielded ores with 13 and 14 per cent. of moisture, but in most cases the percentage is now from 9 to 11. Lately a 7 per cent. moisture on a considerable tonnage was obtained at one mine.

Calcium, manganese and sulphur are negligible. Alumina invariably runs less than 3 per cent. On the South Range, alumina averages less than 2 per cent., there being less interbedded slate here than on the North Range.

Furnace men have repeatedly spoken very highly of the physical character of the ore. The ores as a whole are granular and slightly lumpy, but sometimes platy or even fine and powdery.

North Range ores are prevailing hematitic and red in

color and South Range ores prevailing limonitic and brown. Magnetite occurs sparingly and then only as an admixture in small grains and crystals.

North Range orebodies as compared with those of the South Range are wide and usually quite irregular in shape, due to the greater amount of minor folding, whereas the South Range deposits are longer and narrower and show little or no minor folding. But in both cases the general structural attitude is that of steeply dipping lenses encased in barren rock of various types. Severance of orebodies by igneous dikes is thus far unknown, nor proven, and may be regarded as being absent.

Surface waters encountered in shaft sinking have not been as abundant nor as treacherous as originally anticipated, and now that shafts of a variety of types have been successfully sunk, all fears on that score should be dispelled.

THE MINES.

Thus far I have been presenting only the general considerations needed to give the members of the Institute a perspective of the entire situation. I shall now review the various properties at present being mined, so that the members will better understand operations at the mines should they visit them. Following these descriptions is a table giving a list of the mines, their locations, the names of the operators and local managers or superintendents, and the character of the operations, listed in the same order as described.

KENNEDY.

This is the pioneer mine in the district. The first shipments were made in 1911, and the total shipments to date amount to nearly one million tons. Mining was originally conducted on four closely parallel lenses of ore, but now the ore is taken largely from but two. The shaft is of the wooden drop type with the main level at 262 feet. The surface averages about 125 feet in depth. There is also one timber shaft. The mine has never been worked to capacity, but could easily deliver 400,000 tons or more per year. The ore is medium to coarsely granular, partly brown and partly reddish in color and as mined runs about 55½ per cent. Parts of some of the lenses of ore contain much disintegrated chert which could be washed out and this is now being considered. The moisture content is low.

MEACHAM.

A circular concrete shaft was dropped through 60 feet of surface and continued to 235 feet, but the operation was discontinued just as the crosscut was started toward the ore-body to the south. This deposit is 1800 feet long, very narrow for a North Range deposit, but parts of it are deep. The ore averages between 58 and 59 per cent iron, and parts of it are low in phosphorus.

THOMPSON.

This property embraces three forty-acre tracts running north and south, which contain two separate and parallel deposits. At first a circular concrete shaft was sunk through about 65 feet of surface between the two deposits and a little ore was mined from each and shipped in 1913. The south deposit was subsequently converted into a pit operation. The north deposit is not now being worked. The ore is brownish red and moderately coarse. The pit is one-quarter of a mile long and exposes a maximum width of formation of about two hundred feet. Much of the upper part of this deposit is very siliceous due to the presence of much chert, but as this material is mostly disintegrated, the ore is being beneficiated successfully by a washing process. One small part of the deposit is slightly manganiferous. Shipments are now being made from both the pit and the washer. The average iron content of all the shipments is about 55 per cent. The washed ores are low in moisture.

ARMOUR No. 1.

A circular concrete shaft was sunk through 65 feet of surface. The main level is at 300 feet and the main sub-level at 200 feet. This property has been idle since the shipment of ore from it in the summer of 1913, but this year the westerly part has been stripped and a small quantity of ore is to be shipped from the pit. The ore is moderately granular and red in color with a brownish tone. The ores shipped averaged about 58 per cent iron. A portion of the deposit has slightly manganiferous material associated with it.

ARMOUR No. 2.

A circular concrete shaft was sunk through about 63 feet of surface. The main level is at 160 feet. This property contains a very large tonnage of commercial ore of 60 per

cent. grade, a part of which could perhaps even be mined as Bessemer. Although incompletely explored, this deposit apparently contains all the varieties of formations and ores known for the district. The ores shipped have been red with a purple cast and a semi-metallic lustre. They are mostly fine, mixed with some lump, and in grade represent the best ores ever shipped from the district.

CROFT.

A large circular concrete shaft has been recently ledged, having penetrated 105 feet of surface. Sinking is still in progress. The plan is to crosscut southwardly to the ore deposit at a depth of 200 feet. This is the only Bessemer deposit in the entire district. It contains ore very high in iron, some analyses being over 69 per cent. It is of purple cast with a metallic lustre. The deposit is narrow, one-half mile long and has great possibilities with depth. At one place 60 per cent ore is known to a depth of 380 feet.

PENNINGTON.

This was the first property to be stripped, and the work shattered numerous predictions and established records for stripping. In less than a year's time about 1,000,000 yards were moved and 100,000 tons of ore shipped. The pit is about 1,000 feet long and exposes a maximum width of rock formation of about 400 feet. All during last year and up to August 1st of this year, this mine lay idle, but before the 1915 season closes about 100,000 tons are to be shipped. The ore deposit is a direct continuation of the deposit in the Armour No. 1 and the ores are identical.

QUINN (MAHNOMEN MINING Co.)

This property is now being stripped for pit operation. The surface averages about 65 feet in depth. The pit will be about 1,400 feet long and will expose a maximum width of ore of 200 feet. This deposit is not connected with any other adjacent one. It is located along the southern border of an area that contains an immense tonnage of manganiferous material and itself contains on its south side, or hanging wall, a very large tonnage of it. The commercial ores are reddish in color and will probably average 57 to 58 per cent. when shipped. This mine has great possibilities of ores at depths in excess of 600 feet.

IRONTON.

This mine was opened by a wooden lath shaft. The surface is about 65 feet deep. The deposit is a continuation of the Armour No. 2 deposit and the ores are identical in every respect. About 50,000 tons were shipped last year, but the mine is idle this year.

CUYUNA-MILLE LACS.

This mine was opened by a lath shaft in a surface of about 50 feet. The property contains practically only manganiferous material. The operator lists his ore in four grades based on manganese content: (a) 20 per cent. dried and over, (b) 15 to 20 per cent., (c) 10 to 15 per cent., (d) 10 per cent. and under. The iron ranges from 37 to 40 per cent. dried, phosphorus 0.071 to 0.108 per cent., silica 9 to 21 per cent., moisture 10 to 11¼ per cent. The structure of the ore is good, but the economic conditions governing the use of these ores and the present annual consumption presage a limited output of such material. About 50,000 tons have been shipped up to this year. The mine started operations this year about August 1st and 50,000 tons are expected to go forward.

HILLCREST.

Stripping by hydraulic methods was started last spring and is still in progress. The surface is about 65 to 70 feet deep. The pit will be about 1,200 feet long and expose a maximum width of ore of about 400 feet. The orebody is co-extensive eastwardly with a very large explored but undeveloped tonnage which can also be wrought by pit-mining methods. The Hillcrest ore averages about 57 per cent iron, and as the deposit is compact, operations should prove profitable.

ROWE.

This is the largest pit in operation in the district. The maximum length of iron formation exposed is about 1,200 feet and the width exposed nearly 400 feet throughout. Most of the overburden was removed by hydraulic methods and the remainder by steam shovel. The tonnage of iron formation material that will be handled runs into large figures, but much of it must be beneficated to make a usable ore of it because it contains much disintegrated chert and quartz. This, however, will wash or jig out, and a well-equipped but simple

concentrating plant of small proportions has been built. The better ores have a good physical structure, are brown and red in color and are inclined to be siliceous. About 80,000 tons were shipped last year. At this writing it seems that ore will be shipped before September 1st.

IRON MOUNTAIN.

This property was opened with a lath shaft, the surface being about 60 feet deep. The shaft was sunk to moderate depth and some drifting has been done. The shaft is located on the ore deposit, which is mainly of manganiferous material. About 500 tons were shipped late last fall and it is generally understood that a larger quantity will be shipped this season. This mine portends to produce only manganiferous ores.

CUYUNA-SULTANA.

This property is still in the prospect class and is another of those whose principal problem is making usable a manganiferous iron-bearing formation. Two small exploration shafts were sunk through about 50 feet of surface. One of these is now rigged with a simple headframe and equipped with light machinery to raise enough material for tests and investigations of the ore. The parties interested in the property have for some time been attempting to perfect an electrical method for concentrating and mechanical grading. The surface is shallow and the deposit large enough to warrant stripping, but whether the occurrence of the material is such that a shovel operation is preferable remains to be determined.

ADAMS.

A circular concrete shaft was sunk through 123 feet of surface, a 200-foot crosscut driven at a depth of about 200 feet and a drift cut through the ore deposit for about 500 feet. At this depth the underground work developed a width of ore of 150 feet. The ore is granular and somewhat platy and dark brown in color. The iron content was found to be considerably higher than had been indicated by the earlier drilling. Much of the known ore will average over 58 per cent. About 5,000 tons were stockpiled while development was in progress. The mine was shut down last fall and has been idle ever since. Resumption is contemplated this fall.

WILCOX.

A timber drop shaft was ledged at 93 feet and extended to a depth of 160 feet, a crosscut driven for 80 feet, a main level and a sub-level driven and a cargo of ore shipped, all in the short time of one year. About 50,000 tons are expected to go forward this year. The first 10,000 tons averaged 59 per cent. in iron and much ore of this grade can be mined. The ore is brown and red in color and rather coarsely granular. Up to the present time this ore deposit is the most thoroughly and most systematically explored South Range property, is 4,800 feet long, has a maximum width of 60 feet and a known maximum depth in one place of 300 feet.

BRAINERD-CUYUNA.

A drop shaft was sunk through 90 feet of surface and a main level crosscut driven at 150 feet. The orebody has just been penetrated for 60 feet and drifting is now under way. The first ore hoisted is brown and presages good structure. The property has been only partly explored, but what drilling has been done has indicated that at least a 56 to 57 per cent. ore could be mined.

ROWLEY.

A rectangular concrete shaft is now being sunk. The shaft is not yet through surface, which is 98 feet deep. The drilling disclosed a typical South Range deposit. This property should not be confused with that formerly known as the Barrows mine. The latter is located one-half mile northward and has been idle for over a year.

APPENDIX.

LIST OF CUYUNA MINES AND OPERATORS.

Mine Name.	Description.	Operator.	Superintendent.	Operation.
NORTH RANGE.				
Kennedy	29 and 30, 47-28..	Rogers-Brown Ore Co.....	G. A. Anderson.....	Underground
Meacham	11 and 12, 46-29..			Underground
Thompson	2 and 11, 46-29..			Underground
Armour No. 1	10, 46-29..	Inland Steel Co.....	W. E. Wearne.....	and Pit
Armour No. 2	11, 46-29..			Underground
Croft	1, 46-29..	John A. Savage & Co. (Merrimac Mining Co.).....	John A. Savage	Underground
Pennington	10, 46-29..	Tod-Stambaugh Co. (Pennington Mining Co.).....	John S. Lutes.....	Pit
Quinn	10, 46-29..	Mahnomen Mining Co.....	C. K. Quinn	Pit
Ironton	11, 46-29..	American Manganese	Wm. Pascoe	Underground
Cuyuna-Mille Lacs	3, 46-29..	Mfg. Co.....		Underground
Hillcrest	9, 46-29..	Hill Mines Co.....	Wilbur Van Evers .	Pit
Rowe	17 and 18, 46-29..	Pittsburgh Steel Ore Co.....	John C. Barr.....	Pit
Iron Mountain	33, 47-29..	Iron Mountain Mining Co...N. B. Roscorla.....		Underground
Cuyuna-Sultana	3, 46-29..	Cuyuna-Sultana Iron Co.....	A. W. McGuire.....	Underground
SOUTH RANGE.				
Adams	30, 46-28..	Cuyler Adams	C. C. Adams	Underground
Wilcox	13, 45-30..	Canadian-Cuyuna Iron Mining Co.....	A. A. McKay.....	Underground
Brainerd-Cuyuna	36, 45-31..	Brainerd-Cuyuna Mining Co..C. C. Jones		Underground
Rowley	16, 44-31..	Barrows Mining Co.....	C. B. Rowley	Underground

SOME ASPECTS OF EXPLORATION AND DRILLING ON THE CUYUNA RANGE.

BY P. W. DONOVAN, BRAINERD, MINN.*

Exploration and drilling on the Cuyuna present a few features peculiarly characteristic of the range and a brief consideration of these may be of interest.

PRELIMINARY MAGNETIC EXAMINATION.

A preliminary magnetic examination has an important bearing on the location of holes in spite of the statement frequently made by disappointed explorers that the magnetic lines have nothing to do with the presence of ore. Year by year, the results of exploration, especially on the South range, have increasingly shown the importance and desirability of careful and detailed work of this kind, and while the presence of a magnetic line is not an invariable indication of an ore-body, the fact remains that the lines of maximum attraction constitute the great guide to exploration on this range.

The first step, then, in the exploration of a normal Cuyuna property is a magnetic survey of it to determine the course of the maximum attraction upon it. Or if there is no attraction on it, the course of the trend of the maximum as indicated by its position at the nearest points on each side. This magnetic data will naturally be correlated with drilling or mining information on neighboring properties where it is available.

METHOD OF EXPLORATION.

The method of exploration commonly followed opens with the running of a base line across the property following as closely as possible the course of the maximum attraction. From it holes are located in cross sections at right angles to and at regular intervals along the strike. The normal foot-wall member for the district is the magnetic slate and the

*Local Representative, E. J. Longyear Co., Brainerd, Minn.

normal dip is to the southeast. For these reasons the ore is usually to be expected on the south side of the maximum attraction, the distance varying more or less from place to place. The dip varies from 55 deg. southeast to vertical, 70 deg. probably being the average for the south range, with something a little flatter for the north range. In a few cases dips to the northwest have been found. The first hole on a cross section would be started from 50- to 150-ft. southeast of the maximum attraction and angling towards it. This distance and whether the angle should be 60 or 70 deg. would depend on the depth of surface expected. The position in which this first hole cut the formation would determine the location of the other holes on the same cross section. For the normal south range orebody three holes to a cross section will block it out in sufficient detail. In fact under uniform conditions alternate sections of three and two holes can be used. On account of the greater width of north range orebodies a larger number of holes to a cross section may be required.

Along the strike 300 ft. is the common interval between cross sections, making five cross sections to a forty. Assuming the extension of the orebody the full width of the forty the plan of exploration outlined above would block it out in a manner to permit an accurate estimate with the drilling of twelve to fifteen holes. The average depth of these holes would be about 260 ft., making a total of about 3,400 ft. per forty.

DEPTHS AND KINDS OF SURFACE.

The depth of surface varies from a minimum of 14 ft. in the N. W. part of T. 46, R. 29, to a little over 300 ft. at some points on the east end of the range in Aitkin county. Over the productive part of the south range the average may be said to be about 100 ft. and that for the north range about 80 feet.

The kinds of surface vary considerably from place to place on the range but roughly they may be grouped into three general classes: (1) all sand; (2) gravel, hardpan, boulders and sand, and (3) clay. The last is the least common though it is found in the northern part of T. 47, R. 26, in Aitkin county.

These various characteristics are of special interest in their bearing on shaft sinking or stripping. The all sand surface

while offering serious difficulties for shaft sinking, is ideal for handling by steam shovel or hydraulic methods where other conditions make stripping possible. Illustration of the three methods of opening in this type of surface are the Barrows shaft in Sec. 10, T. 44, R. 31, on the south range; the Armour No. 1 steam shovel pit in Sec. 10, T. 46, R. 29, on the north range and the Hill Crest hydraulic pit in Sec. 9, T. 46, R. 29. The Wilcox, an expeditiously sunk drop timber shaft in Sec. 13, T. 45, R. 30, on the south range, went through 91 ft. of surface, the upper 65 ft. being gravel and the last 26 ft. clay. In general it can be said that at no place on the range has been found any such succession of boulders as are sometimes encountered on the Mesabi range, and the difficulties arising from such a condition are not to be contended with here either in drilling or stripping.

On account of the variety in surface conditions the preservation of surface samples in drilling is of first importance. The conditions they indicate may have much to do with the choice of the method of opening to be used and a little attention to this matter during the first drilling will largely obviate the necessity of special surface test holes when opening is under consideration. In the same connection, all possible data as to water level should be secured while drilling is in progress, as this information, correlated with the observation of the surface samples, will throw much light on the conditions to be expected in shaft sinking.

DRILLING PRACTICE.

The outfit used is the light churn drill equipment with separate diamond drill attachment as developed on the Mesabi range. Its adaptation to angle hole drilling, particularly for surface or churn drill work, has been largely a local development. In this respect the chief feature of interest is the use of two auxiliary legs with the tripod. They are set at the angle of the hole to be drilled and tied to the front of the tripod. A movable cross piece slides up and down on them and takes the weight of the casing as well as holding it to the proper angle. This feature saves much time in setting up, strengthens the tripod for heavy surface work and greatly facilitates the drilling operation.

The crews themselves, originally recruited from the Mesabi range and experienced in vertical hole drilling, have shown commendable ability in adapting themselves to conditions

here. In the early days on the range, 1905 and 1906, much difficulty was encountered in driving the 3-in. casing through surface in angle holes, and not infrequently it would be hopelessly stuck at depths of less than 100 feet. In more recent practice many angle holes have been driven through as much as 250 ft. of surface and an average of 15 ft. or over per shift maintained for the whole distance.

VERTICAL OR ANGLE HOLES.

There has been considerable discussion of this question in the technical journals and a detailed consideration of it is outside the scope of this article. It must be obvious, however, that for the conditions existing on the south range, angle holes are essential. The greater width and flatter dip of north range orebodies permits a wider use of vertical holes but even there they should not be used exclusively. One angle hole to a cross section or alternate sections of vertical and angle holes will give more complete data as to the character of an orebody than vertical holes alone.

The outstanding structural feature of the Cuyuna formation is the close stratification, both of the ore lenses and the enclosing walls and it is evident that that kind of hole, which, for a given footage, cuts the largest number of these strata will be the best from an exploratory standpoint; and within the depths which are used for 90 per cent of the holes there should be no difference in the samples from angle and vertical holes, for identical methods are used in drilling them.

One point should always be borne in mind, however, in the comparison of results from angle and vertical holes, and of drill samples and mine samples. That is, that on account of the stratified structure a 5-ft. sample in a drill hole does not represent the ore in a 5-ft. horizontal plane encircling the hole as it would in a massive Mesabi orebody, but in the 5-ft. (more or less) plane conforming to the dip and strike of the stratum or strata through which it has passed. Thus a vertical hole might continue a considerable distance in one narrow but steeply dipping stratum which might represent conditions quite different from those on either side of it at right angles to the strike.

DRILL AND MINE SAMPLES.

As far as the development of the district has gone the drill hole samples and subsequent mine samples on the same prop-

erty may be said to have checked very closely, in most cases the mine samples running one-half to one per cent. higher than the drill samples. Considerable has been said as to the mine samples from some of the manganiferous orebodies running uniformly 6 to 12 per cent. higher in manganese than the drill samples in the same orebodies. It is doubtful, however, whether systematic work in sufficient detail has really been done to establish such a fact. There would seem to be no reason why a carefully taken drill sample in a manganiferous orebody should not be as representative of the material passed through as a similarly taken sample would be in an iron ore. One fact to be borne in mind, however, in the consideration of this question is that the most striking characteristic of the manganiferous orebodies is the extreme irregularity of the manganese content. With this in mind it can readily be seen that a drill hole in such material may not be representative of material for any distance around it, even though correct and accurate for that through which it has passed. For this reason a manganiferous orebody will require a greater number of holes in a given area to show it up accurately than would an iron orebody of the same area.

HARDNESS OF THE ORE AND IRON FORMATION.

The greater part of the ore in the district, but especially on the south range, is soft enough for churn drilling. The iron formation on the south range is also soft so the total proportion of diamond drilling is small. On a typical south range property consisting of several forties the diamond drilling was 15 per cent. of the total. If the surface drilling be excluded and only the ledge considered the diamond drilling was 32 per cent. and the churn drilling 68 per cent. A typical north range property on the total showed 33 per cent. diamond drilling and 67 per cent. churn drilling; ledge drilling on the same property was 45 per cent. diamond drilling and 55 per cent. churn.

CARBON LOSS.

As might be expected, the slates and schists and even hard ores of the south range give a comparatively small carbon loss. On the other hand, the cherty character of much of the north range formation and the frequent quartz seams, give quite a different condition. In some of the ferruginous cherts and cherty hard ores one bit will be good for only two or three feet and the carbon loss is relatively high.

GENERAL.

As one studies the record of intelligently directed Cuyuna explorations the one feature which perhaps stands out above all others is the small number of wasted holes. The magnetic lines forming a basis for the location of the first holes and the regularity of the trend of the formation, make it possible to place almost all of the holes in the ore formation. A compilation of the exploration records on six developed orebodies on the south range shows an actual total of 3,000 ft. of drilling per forty. This compares with the 3,400 ft. per forty arrived at theoretically in an earlier paragraph of this paper. Each foot of this drilling developed 250 tons of merchantable ore. That is, at the average rates for drilling which have prevailed on the range for the past 5 years, the exploration cost of developing 1 ton of ore was 1 cent. While the record for the entire range would probably be a little higher than this we believe that the Cuyuna showing in this particular will compare favorably with any other district in the Lake Superior region.

In addition to this inducement for exploration there is no question that the range has possibilities for orebodies now unsuspected, in parallel or displaced lenses. Within the last year there have been several cases where a careful consideration of apparently insignificant magnetic indications has led to the discovery of important orebodies, and it is the opinion of those who have given the district the most careful study that the future has in store many similar results. Thus the greatest possibilities for ore in unexplored areas are on lands close to and parallel to the present outlined orebodies rather than on lands in newer and more distant areas.

ROCK DRIFTING IN THE MORRIS-LLOYD MINE,
THE CLEVELAND-CLIFFS IRON CO.

BY J. E. HAYDEN, ISHPEMING, MICH.*

The Morris-Lloyd mine of The Cleveland-Cliffs Iron Co., is located at North Lake, four miles west of the City of Ishpeming. At a distance of 3,000 ft. east of the Lloyd shaft, an orebody was discovered by diamond drilling from surface. In the fall of 1914 it was decided to open up this orebody by drifting from the Lloyd shaft on the 600-ft. level. A 9- by 10-ft. heading, without timber, was driven due east from a point 350 ft. south of the shaft in the slate footwall, which strikes almost due east and west, and dips at an angle of about eighty-five degrees to the south.

Progress—The drift was started on September 1, 1914, and completed on June 17, 1915, a distance of 2,960 ft. having been driven, in 240 working days. The record month was May, 1915, when 406 ft. of drift was driven in 26 working days. During this month the best day's progress was 19 ft. 9 in., (due to a partly missed-cut on the previous shift); the poorest day's was 12 ft. 9 in., the average daily progress being 15 ft. 8 inches. The average monthly progress throughout the 9½ months was 311 ft. 6 in., the average daily progress being 12 ft. 4 inches.

The materials encountered were slate, greywacke and quartzite. The work was done on two 8-hour shifts, 4 miners, and, for the greater part of the time, 6 muckers constituting the crew. The muckers alternated, one gang of 3 filled a car, while 3 rested. Two No. 18 Ingersoll-Leyner drill machines were used throughout the work, except for the first 100

* Mining Engineer, The Cleveland-Cliffs Iron Co.

ft., when two $3\frac{1}{4}$ -in. piston machines were used. Electric motor haulage was used in 2,300 ft. of the drift, and hand tramping in the first 600 feet. Permanent track on a $\frac{3}{4}$ per cent. grade was laid and kept up to the face of the muck-pile. The trolley wire was kept up within 150 ft. of the breast, the muckers tramping the car this distance to the motor. At intervals of 300 ft. along the drift, sidings were cut to

DIAGRAMS TO ILLUSTRATE CUTS USED IN DRIFTING AT THE MORRIS-LOYD MINE

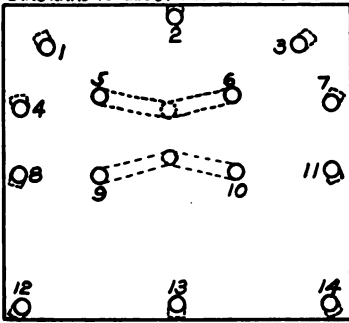


FIG. 1.

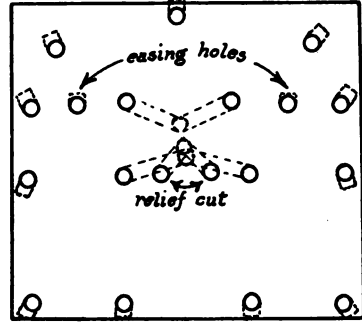


FIG. 2.

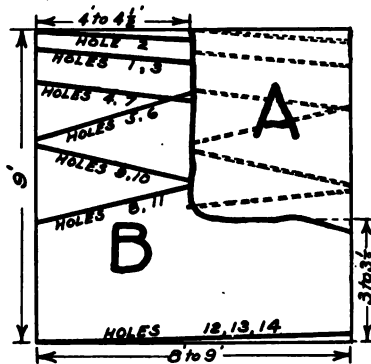


FIG. 3.

hold three motor cars. Throughout the work, 80 per cent. nitro-glycerine dynamite was used. The fact that the drift paralleled the slips, made it necessary to use a strong explosive to insure breaking the cut, as the ground was extremely tight. This also eliminated large chunks and threw the muck back a considerable distance from the breast, enabling the miners to quickly rig-up for drilling the next cut.

Cuts Used in Drift—As the ground varied considerably during the progress of the drift, it was necessary to vary the cut used somewhat. Fig. 1 and 2 represent the two cuts used. Fig. 1 was the cut commonly employed for the slate drifting, and Fig. 2 for the quartzite and greywacke. This latter cut was varied as the ground required.

In Fig. 1, holes 1 to 11 were drilled from the bar in one position, and fired in one blast. To complete the square, all 14 holes were fired on the second blast. It was found that only when holes 5-6 and 9-10 intersected at their respective ends, that good results were obtained. This point was carefully watched, the shift boss always testing these holes before each blast. For the extremely hard ground, such as the quartzite encountered, the cut was made as in Fig. 2, the top 15 holes being fired in the first blast, and all 19 holes to complete the second cut and square. The two relief cut holes, (see Fig. 2), were drilled to meet at a depth of 3 feet. For the moderately hard ground, such as the greywacke, two easing holes were drilled as in Fig. 2, instead of the 3-ft. relief cut.

Fig. 3 shows the cycle of operation to complete the square, "A" representing the portion removed in the first blast, "B" that portion removed in the second blast. Each shift blasted twice, completing the cut, leaving the drift squared at the end of the shift. During the entire progress of the drift, there was but one slight injury, that being caused by a piece of rock hitting a miner's hand while barring down loose ground from the back.

Disposal of Rock—Three-ton saddle back motor cars were used in the work, having a height of 5 ft. 6 in. above the rail in the clear, which necessitated considerable lifting by the muckers. This difficulty was greatly overcome by the installation of a portable loader in the latter half of April. This loader consisted of an inclined steel-apron conveyor belt with 2¼-in. angle irons placed 15 in. center to center, mounted on a roller bearing truck, operated by a 5-h.p. series motor taking current from the underground haulage wire. The loader was arranged so that the angle of the incline could be reduced, allowing it to pass under the trolley wire. The muckers shoveled into a hopper at the lower end of the belt, the rock being conveyed up the incline and dumped into the motor car, which was run under the top end. A small part of the rock could be picked down from the pile into the hop-

per. Three muckers shoveled, while one trimmed the car. The work was speeded up after the loader went into commission, as it became possible to handle more rock. Within a week the output was increased 25 per cent., and later on it often reached 30 per cent. The whole cycle of operation was completed earlier on the 8-hour shift, giving more time for drilling, consequently deeper cuts were tried, with extra holes, to insure breaking.

Ventilation—At a distance of 1,220 ft. from the shaft, a fan-station 12- by 14-ft. was cut, and a No. 10 Buffalo Forge



PORTABLE LOADER, INSTALLED TO FACILITATE THE LOADING AND DISPOSAL OF ROCK

Co. steel-pressure fan of 20,000 cu. ft. per min. capacity installed, capable of operating either as a suction or a blower. This was operated by a 240-volt, 15-h.p. direct-current motor, taking current from the underground haulage wires. Ten-in. riveted steel pipe was connected to this fan, one end being kept about 75 ft. from the breast, the other discharged into the pipe compartment of the shaft.

After a blast, the fan was started as a suction from the breast, and at the same time a small jet of compressed air was allowed to escape in the breast. The compressed air forced

the smoke and gases down the drift to the suction end of the fan-pipe. Without this compressed air blowing in the breast, it was found that some of the smoke and gases got by the suction end, which was placed on the rib-line at a height of 6 ft. from the floor. After 15 min. of operation as a suction, the fan was reversed, and fresh air blown in the breast, when work could be resumed with safety. No artificial ventilation was used in the 1,220 ft. of drift up to the fan-station. As most of this work was done during the cold weather when the shaft was strongly up-cast, the blowing of air in the breast was sufficient to clear the drift in 30 minutes.

Progress By Months—September, 1914: The first half of the month the drift advanced 100 ft., using $3\frac{1}{4}$ -in. piston machines; in the second half of the month, with Water-Leyners, it advanced 150 feet.

October, 1914: The drift advanced 325 ft. in 26 working days.

November, 1914: The drift advanced 270 ft., and two crosscuts were started in 25 working days. Up to this time 4 muckers and hand-tramming had been employed. With the installation of the motor haulage in the latter half of this month, the mucking force was increased to 6 men.

December, 1914: Advanced 321 ft. in 23 working days.

January, 1915: Advanced 298 ft. in 24 days. Throughout the month the drift was in hard quartzite. A cut required from 80 to 100 bits; to complete a square required from 100 to 150 bits. An average of 180 to 250 bits were used per 8-hour shift.

February, 1915: Drifted 252 ft. in very hard quartzite in 23 working days.

March, 1915: An advance of 252 ft. in quartzite in 27 working days.

April, 1915: The first half of the month the drift advanced 153 ft., the second half 187 ft., or a total of 340 ft. in 26 working days. During the latter half of this month the portable loader was used.

May, 1915: Was the record month, the advance being 406 ft. in 26 working days, or a daily progress of 15 ft. 7 inches.

June, 1915: Advanced 205 ft. in 15 working days, completing the drift.

The following is a detailed cost of the 2,960 ft. of drift:

	Labor.	Supplies.	Total.	Cost Per Ft.
Miners	\$ 8,430.27	\$ 1,587.73	\$10,018.00	\$ 3.384
Mucking	8,169.03	8,169.03	2.760
Tramming	1,117.92	498.39	1,616.31	0.546
Explosives	6,889.86	6,889.86	2.325
Air	720.00	720.00	0.244
Shop expense	460.00	230.00	690.00	0.233
Machine repairs	135.50	1,196.27	1 331.77	0.451
Air and water hose....	75.00	75.00	0.025
Carbide	95.00	95.00	0.032
Total cost	\$18,312.72	\$11,292.25	\$29,604.97	\$10.000
Per foot	6.18	3.82		

THE MINING SCHOOL OF THE CLEVELAND-CLIFFS IRON COMPANY.

BY C. S. STEVENSON, ISHPERING, MICH.*

The Mining School of The Cleveland-Cliffs Iron Company is of that class of trade schools known as Industrial Corporation Schools, the purpose of which is the mental improvement of those already enlisted in the industry. There are but a very few of this general type in the United States and each is operated on a plan peculiar to local conditions, the one thing in common being that the work taught is in harmony with the industry concerned. A great many such are operated in Germany and by many they are credited as being largely instrumental in producing the great industrial development of that country during the past 20 years.

The Purpose of the Mining School—It is essentially true that the foreign labor which has been absorbed in large numbers by our mines in recent years is an inexperienced product. It is, however, not the purpose to attempt to teach these men (except in unusual cases) since by difference in language and a lack of early education they are not amenable to school work of this character. The prime function of the school is to train to the highest possible degree of efficiency the English speaking men upon whom this inexperienced foreign product depends for its guidance. The school, therefore, is not open to all underground employes of the company but concerns itself only with a group of men who are carefully selected by the superintendents and mining captains on a basis of their ability and mining aptitude.

Before instituting the work a serious effort was made to locate and study the method of operation of similar schools so that the common elements of these might be taken as a frame-work around which our instructional work might be constructed. This investigation proved that there were none in the United States the aims and purposes of which were at

*Director Educational Department

all similar to the one we proposed to establish. Investigation did, however, indicate that a school in its ordinary sense and our Mining School should be very dissimilar in their aims and purposes. The public ones have for their purpose a broad, mental and cultural development. The Mining School, on the other hand, while not ignoring the desirability of such instruction, largely disregards the curriculum and methods thereof and concerns itself wholly with instructional work calculated to increase the workman's efficiency and co-incidentally his earning capacity. In short, it is designed to have a definite value in dollars and cents, not only to the miners who participate in the work, but to the company as well.

Attitude of the Men Towards the School Work—In the beginning it was noted that the men were as a rule indifferent, if not antagonistic. Attention, however, should be directed to the fact that a few men of especial ambition and energy welcomed it, several of whom had already attempted to help themselves through the medium of the correspondence schools. Some, however, looked upon the work with a suspicion that it was intended to benefit the company and not themselves. They felt that their minds and bodies were in a rut and that the company was arrayed against them. Gradually, but not without difficulty, these prejudices were broken down and replaced with a spirit of open-mindedness and enthusiasm. The company has authorized the statement that in so far as possible all men chosen for shift bosses will be taken from the ranks of the Mining School. This gave the men a definite motive for attendance and interest and assisted greatly in quickly breaking down all prejudices, since it proved that the work was an undertaking of mutual concern to the company and miners as well. On June 1, 1915, the work of the first class, comprising 33 men, was completed, and it can be stated definitely that for the greater period of their course the men manifested a higher degree of open-mindedness and enthusiasm than is usual in high schools and universities.

Time Given to the Course—The students enrolled in our school are largely men with families and ordinarily quite a large portion of their leisure time is given to domestic affairs. The school intrudes on this and it would be unreasonable to suppose that the men would willingly sacrifice this time from their home affairs for a long period. For this reason the work of a single class is designed to cover one and a half years. This length of time proved, if anything, too short for the

instruction in the subjects covered by the course but this was overcome by the simple expedient of increasing the length of class periods and also the amount of home preparation.

Each miner according to our present system attends two classes a week, each of an hour and a half. If the miner is working on the day shift he attends the evening classes and if he is working on the night shift he attends the afternoon sessions. All of the class work is done on the miners' own time and they receive no remuneration from the company for that given to the school work.

*Readiness With Which the Men Acquire Information—*The experience gained with our first class proved that the men can readily assimilate information if care is always taken to bring out the practical application of the instruction to their daily work. For example, a course in Arithmetic would be a failure if taught as an abstract subject but if the instruction is prepared with a view to its practical application to the daily problems of a miner's life, the student is interested and for the first time sees the purpose of the instruction which bored him in his early school days. In short, the power to assimilate information is in direct proportion to the practical value of the instruction. The men have a skill derived from long experience in mining and can perhaps more readily assimilate academic instruction relating to the industry than can the average university student lacking such experience. However in their ability to comprehend abstract information they rank considerably under the students of the high schools and universities.

*Factors Controlling Attendance—*The Mining School of our company began its first class with an enrollment of 38 men and of these, 33 successfully completed the work offered by the department. Four of the five men, who began but did not complete the course, withdrew from the work on account of business conditions, which made their attendance impossible. We are very proud of this record of attendance since we have had a much lower rate of attendance mortality than has been reported by similar schools in the United States. Many devices were resorted to for the maintenance of attendance. First of all, a high degree of personal friendship was established between the students and the instructor. In all cases it must be borne in mind that the men are not children but of mature years and respected in the communities in which they live, and great care is taken not to wound their

pride and self-respect. Infinite patience must be a virtue of the instructor to an even greater degree than is common to the teaching profession. In case a student grows discouraged and fails to attend classes, an encouraging letter is sent to him together with a copy of the instruction paper for the succeeding lesson, and upon his return to the class room an increased amount of personal attention is given until he again feels that he is on a par with the other men. In so far as possible the formal atmosphere of the ordinary class room is avoided and replaced by conditions calculated to make the men feel comfortable and at home. Illustrations in lectures are taken whenever possible from the experience of our own company which gives the men a personal interest in the subject under consideration. In short, a feeling of fellowship and confidence must be created early in the work, after which many problems may be ironed out satisfactorily.

System of Instruction—We have adopted with success what is known as the "Unit Course," in which the entire attention of the men is fixed on one subject until its completion. Experience has proved this system to be much better adapted to our needs than the teaching of several subjects coincidentally.

As a nucleus for each course, instruction papers have been prepared in either mimeographed or printed form. These instruction papers become the property of the men and form a convenient means of reference in the future. They are, however, but a minor part of the instruction, most of which is imparted by lectures.

Development of Independent Thinking—The experience gained with our first class has proved that perhaps the greatest weakness of the men is in their lack of power to do original and independent thinking. To correct this mental condition a series of informal discussions on mining topics was instituted early in the course. These were not a scheduled part of the course and they followed the usual class period, the discussion being led by the instructor. This interchange of ideas broadens and helps the men and is perhaps as large a factor in the production of a man of reliability and common sense as is the pure school work itself. Mention of a few of the topics discussed is given herewith:

Safe methods of blasting down timber.

Methods of thawing dynamite.

The use of delay action fuses in shaft sinking.

The location of holes in blasting various types of ground.

The choice of explosives for different character of ore and rock.

The proper methods of charging and tamping explosives.

The choice of drilling machines for different classes of work.

The elementary features of rock drill construction.

The care and use of rock drills.

The proper methods of setting timber and the variety of timber to use in caps and legs.

The advantages of systematic sub-level work over unsystematic sub-level work.

The relative merits of timbered and untimbered raises.

The proper thickness of a sub-level slice from the standpoint of safety, costs and recovery.

The inspection and lubrication of hoist ropes.

The testing of safety catches on cages.

The cost of producing compressed air.

Underground sanitation.

Ventilation of metal mines.

The sampling of ore and its relation to the marketing of ore.

The proper degree of discipline of the shift boss over the miner.

Methods of procedure at mine fires.

The treatment of a man overcome by powder smoke and other first-aid problems.

The Workmen's Compensation Law.

The proper use of the various report blanks which are filled out by underground employees.

The informal discussions above referred to were valuable but experience proved that a few of the men had hesitancy in expressing their ideas. To reach these men we began our monthly "Suggestion Papers." These involved the preparation by each student of an essay on any mining subject of his own choosing, once each month. In the preparation of these the services and advice of the instructor were freely given. A high standard of neatness and accuracy was demanded. The papers submitted were of an unexpectedly high degree of merit and they indicated a very laudable desire on

the part of the men to do real, independent thinking. They as well had a secondary value in the development of penmanship and in the use of the English language. It can be definitely stated that, as a result of this effort to mentally awaken the men, there was a marked improvement shown in their ability to do original thinking and in their power of analysis.

Age, Nationality and Previous Schooling of the Students—

The average age of our first class was 32 years at the time of beginning the course. The youngest student was 22 and the oldest 50 years of age.

The nationality of the men was as follows:

American born	13
English born	12
Finnish born	4
Swedish born	2
Italian born	2

The average number of years spent in school previous to attending the Mining School was 4.3 years. The range of time previously spent in school varied between two months and 10 years.

What Should Be Taught in a Course of This Character—

Since the time spent in the work is small it is evident that only such subjects should be taught as are of practical value to the student in procuring his advancement. It is better to teach a few subjects thoroughly than to teach a smattering of a large number of subjects. In the choice of these the limited early preparation of the men cannot be ignored and any tendency to introduce university or even high school standards must be carefully avoided. In order that the instruction in a school of this character may be sufficiently effective to justify company approval and subsidy, two principles must be adhered to: first, courses of study should be developed from mining situations and be adapted to mining needs; second, the various employments of the men should be investigated and analyzed in a search for the common elements on which group teaching can be based. The following course was followed by our first class, which completed its work June 1st of this year. It is designed to cover fundamental subjects on which foundation the student can build after he has left the school. Each of the subjects was taught in the order in which it is here named.

1. Arithmetic.
2. Elementary Drawing.
3. Geometrical Drawing.
4. Mechanical Drawing.
5. Geology.
6. Construction and Use of Mine Maps.
7. First-Aid to the Injured.
8. Time-Keeping.
9. Mine Sampling.
10. Mining Methods.
11. Business Correspondence.

DETAILED REVIEW OF THE WORK TAUGHT.

Arithmetic—The instruction in Arithmetic has for its object primarily to impress on the men the necessity for acquiring a thorough system of making, with as much self-dependence as possible, the more simple calculations relating to the wages of miners, costs of mining and estimates. A total of 18 special instruction papers were prepared for and used in this work. These papers were designed, in so far as possible, to cover the needs of the mining industry. The parts of Arithmetic treated were:

- Addition.
- Subtraction.
- Multiplication.
- Division.
- Cancellation.
- Addition of Fractions.
- Subtraction of Fractions.
- Multiplication of Fractions.
- Division of Fractions.
- Addition of Decimals.
- Subtraction of Decimals.
- Multiplication of Decimals.
- Division of Decimals.
- Percentage.
- Proportion.
- Areas of Surfaces.
- Computation of Volumes.
- Powers and Roots.

As indicating the difficulty which we encountered in the instruction in this subject it may be said that no more than five of the men had ever completed a course in Arithmetic and

there were many who in the beginning could not make the simple computations in Addition and Subtraction.

Elementary Drawing—Drawing is the sign language of the machanic. In discussing a practical problem the first thought of the shift boss and the mining captain is to make or attempt to make a sketch. Modern mining development demands that the shift bosses and captains be able to understand and work from blue-prints. For these reasons the subject of mechanical drawing was taught in the Mining School. In elementary drawing five simple drawings were made by each student which served largely to accustom them to the use of drawing instruments and the fundamental principles of making a drawing.

Geometrical Drawing—This course has a two-fold value, first, it serves as a preparatory subject to mechanical drawing and, second, it gives the student a working knowledge of geometrical facts which have many common and practical applications. A total of four drawing plates, covering 24 geometrical principles were required in this course.

Mechanical Drawing—In this subject each student completed five drawings, beginning with simple mechanical devices and proceeding to more complicated work. The prime purpose, which was to teach the men how to read a mechanical drawing, was accomplished. The character of the work done by the men in this subject was of an unexpectedly high degree of merit and closely approached the work of similar nature which is done in universities. The interest which the men took in it was manifested by the fact that the majority of them have purchased mechanical drawing instruments for their own use. A considerable amount of interpretation of blue-prints was required of the students in connection with this work.

The course in mechanical drawing had a secondary value in the development of system and accuracy. In the beginning the men were found to make numberless mistakes in measuring dimensions and in the details of construction. Gradually, however, these faults were overcome and the men accustomed themselves to think and work accurately.

The work taught in Elementary, Geometrical and Mechanical Drawing is based on a printed instruction paper written for the especial needs of the Mining School.

Geology—In so far as possible all instructional work in this subject is based on the Geology of the Marquette range.

A printed instruction paper is used as the nucleus of the course and with our first class this was supplemented by lectures and the study of approximately 150 specimens of rocks and minerals. The men took a very lively and almost unexpected interest in this subject. It was found that some of the men had a fairly good idea of the geology of the range in the beginning and these welcomed the opportunity of perfecting the information which they had gained largely through practical experience. Many of the men, it was learned, have mineral collections in their homes and many specimens of rocks and minerals were presented to the instructor for identification and discussion. It is believed that there is no course more valuable than geology in making a miner's work more interesting and less of a drudgery. The course followed the following outline:

DYNAMICAL GEOLOGY.

1. The effect of the atmosphere on rock formation.
2. The decay of rocks.
3. The formation of sedimentary rocks.
4. Aqueous agencies.
5. Mechanical effects of water.
6. The formation of water falls.
7. The eroding power of streams.
8. The formation of deltas.
9. The action of glaciers, especially on the geology of the Marquette range.
10. Chemical effects of water.
11. Chemical deposits from springs.
12. The condition of the interior of the earth.
13. The effects of heat on rock formation.
14. Organic agencies.
15. The formation of coal and limestone.

STRUCTURAL GEOLOGY.

1. Exposures of rock available for study.
2. Definition of the term "Rock."
3. Classes of stratified rocks.
4. Dip of rocks.
5. The outcrop of rocks.
6. Anticlines, Monoclines and Synclines.
7. Conformability of rocks.
8. Fossils.

9. Igneous rocks.
10. Igneous rock classification.
11. Metamorphic rocks.
12. Structure common to all rocks.
13. Joints in rocks.
14. Fissures.
15. Normal faults.
16. Reverse faults.
17. Forms of orebodies.
18. Definition of "Ore."
19. Discussion of the form in which orebodies occur on the Marquette range.

HISTORICAL GEOLOGY.

1. Discussion of the geological section.
2. Discussion of the succession of rocks on the Marquette range.
3. Detailed description of the rocks of the Marquette range—illustrated by specimens.

HISTORY OF THE MARQUETTE RANGE.

1. Date of discovery and record of development.
2. History of the Swanzey range.

IRON ORES.

1. Discussion of the composition and the characteristics of various iron ores.
2. Discussion of the ores of the Marquette range.
3. The use of the dip needle in the location of orebodies.
4. The occurrence of soft and hard ores.
5. A detailed description of the ore deposits at the Maas, Negaunee, Austin, Stephenson, Lake and Cliffs Shaft mines.

The Construction and Use of Mine Maps—In this course it is desired to teach the fundamental details of map construction with a view to facilitating the student's interpretation of the maps supplied to him by the Engineering Department. The experience of our Engineering department indicates that there is a definite need for instruction of this character. The course is based on a mimeographed instruction paper which follows the outline given below:

- A. The Use of a Compass.
 - 1. Description of compass.
 - 2. Degree of accuracy secured in compass work—magnetic attraction.
 - 3. The reason for reversing the east and west points of a dial on the compass.
 - 4. The method of procedure in using a compass.
 - 5. Problems illustrating the use of a compass in sub-level work.
- B. The Use of a Clinometer.
 - 1. Determination of the angle for putting up a raise and use of the lines given for a raise by the engineers.
- C. Templates for Track Curves.
 - 1. The grades of tracks.
 - 2. Use of a hand level and track level.
- D. Description of the Protractor and Engineers' Scale.
- E. The Construction of Maps.
 - 1. Coordinates.
 - 2. The relation of the coordinates of a sub-level to those of the sub-level above and below.
 - 3. The scale of mine maps with sufficient problems to enable the student to take distances from maps.
 - 4. The zero point or origin of a survey.
 - 5. Government surveys.
 - 6. The explanation of the use of cross-hatching in constructing mine maps, also coloring.
 - 7. Problems in mine mapping.
- F. Mine Levels.
 - 1. Sea level datum.
 - 2. The use of an arbitrary datum plane.
 - 3. The proper use of elevations, supplied by the engineers, at the top of each raise. The disadvantages of having sub-level drifts meet off level.
- G. General Considerations in the Use of Mine Maps.
 - 1. Systematic sub-level work in relation to efficiency in handling timber and supplies.
 - 2. The relation of systematic sub-level work to maximum recovery of ore.

3. The relation of systematic sub-level work to the safety of miners and its relation to the ventilation of a sub-level.
4. Procedure in locating a block of ore which has been lost on the sub-level above.
5. Assay maps and their use.
6. Use of maps in holeing into or connecting with other workings, whether abandoned or where men are at work.

First-Aid to the Injured—The work of first-aid to the injured has taken a place of such importance in mining that it would seem unnecessary to elaborate on the reasons for including instruction thereon in a course of this character. This work was given through the medium of lectures, which followed the following outline:

1. The history of first-aid work and its aims and purposes.
2. The structure of the body.
3. Description of the various types of bandages used in first-aid work.
4. Description of wounds and prevention of infection, the treatment of shock and the use of stimulants.
5. The circulation of the blood and the control of hemorrhage.
6. Bruises, sprains, dislocations and burns.
7. The treatment of fractured bones.
8. Respiration and the standard methods of inducing artificial respiration.

Time-Keeping—In view of the fact that our company is selecting its new shift bosses from the ranks of the Mining School it is important that the students be instructed on the methods of time-keeping. This subject was presented to the students in the form of a mimeographed instruction paper which explains in detail the system of time-keeping used by The Cleveland-Cliffs Iron Company.

Mine Sampling—The instruction in this subject followed the outline given below:

- A. Theory, and importance of close attention.
- B. Methods of application in use by The Cleveland-Cliffs Iron Company in:

1. Drifts.
2. Stopes.
3. Raises.
4. Mine cars.
5. Skips.
6. Railroad cars.
7. Steam shovel loading.
8. Stockpiles.

C. Treatment of samples.

1. Labelling.
2. Crushing and drying.
3. Quartering.
4. Bucking-down.

Mining Methods—This subject was presented to the students in a mimeographed instruction paper covering the methods of mining common to the iron ranges of the Lake Superior District. The course followed the following outline:

1. General principles governing the selection of a mining method.

- (a) Open cut mining.
- (b) Steam shovel mining.

2. Method of mining medium and hard ores.

- (a) Milling.
- (b) Underhand stoping.
- (c) Back stoping, Case 1 and 2.
- (d) Block caving, Case 1 and 2.
- (e) Sub stoping.

3. Method of mining soft ores.

- (a) Room and pillar square set.
- (b) Room and pillar square set using filling.
- (c) Top slicing one set high.
- (d) Top slicing two sets high.
- (e) Sub caving, Case 1 and 2.

4. Detailed description of the methods of mining used by The Cleveland-Cliffs Iron Company in the Negaunee, Ishpeming, North Lake, Republic and Gwinn districts.

Business Correspondence—In view of the students' inexperience in business correspondence it was thought advisable as a final course to instruct them in the art of writing a good business letter. This subject was presented to the students in a mimeographed instruction paper, each student being re-

quired to write at least 12 business letters which were graded by the instructor for neatness and their conformability to established forms and customs in business correspondence.

Instruction of Mechanics and Electricians—This paper would not be complete without mention of the educational work which is being done by our mechanical and electrical departments. In this work engineers act as instructors and any employe engaged in mechanical or electrical work is privileged to attend the classes. Evening classes only are held, the men receiving one lesson each week. The work is very practical in its nature and excellent results have been obtained.

This paper is presented as a record of what has been accomplished thus far by the Educational Department of our company. We realize that the plan here presented can be improved upon and certain improvements are already under consideration. Whether or not the school is permanent may be safely left to the future. At present it meets an urgent need and will until cooperation with the public schools can be effected.

HYDRAULIC STRIPPING AT ROWE AND HILLCREST MINES ON THE CUYUNA RANGE, MINNESOTA.

BY EDWARD P. M'CARTY, MINNEAPOLIS, MINN.*

The Pittsburgh Steel Ore Company in 1913 introduced, at the Rowe mine, the hydraulic method of removing overburden on iron ore deposits. Hitherto, the use of the steam shovel had been considered the most satisfactory method of doing such work. Other methods tried at different times had invariably resulted in failure. The use of water at the Rowe and Hillcrest mines was not only feasible but also economical because of the location of the orebodies and the character of the overburden. Reference to Plate 4 shows in plan the orebody and vicinity at the Hillcrest mine. Conditions quite similar prevail at the Rowe mine where the top of the overburden lies at considerable elevation above the water and the top of the ore is about 20 ft. below the water. The ore and the pit are now protected from flooding by a clay dike.

The Rowe mine is adjacent to Little Rabbit Lake, where the water pump, with a capacity of 3,500 gal. per min., was placed. The water was pumped through about 1,500 ft. of 12-in. pipe to the point chosen for excavation. Here the pipe was reduced and an ordinary hydraulic giant was fitted. The size of the giant nozzle was varied for the different materials encountered, but for the average work a 4-in. nozzle was used. The water pressure at the nozzle was about 50 pounds. The stream was directed against the bank and the material was washed down a rough channel to where a 12-in. Morris sand pump was located. The suction of the sand pump picked up all the water and sand material and pumped it out through a 12-in. pipe to the spoil bank. The discharge pipe of the sand pump varied in length from 500 ft. to 2,400 feet. The vertical distance from where the sand pump picked up the

*Professor of Mining, University of Minnesota.

material to where it deposited it on the spoil bank was from 27 to 40 feet.

It was found that the material brought down by the hydraulic giant could be washed to the sand pump on a grade as flat as 4 ft. in 100 feet. By locating the sand pump on a platform in one place the giant was worked all around the pump in a gradually increasing circle until this 4 per cent. slope was reached. With an average depth of 54 ft., this limit was not attained until the giant had swept a circle around the pump of a 1,350-ft. radius. Compared to the constant moving of cars and track for a steam shovel outfit, this made quite a saving.

For a short time at the beginning of the operation a plung-

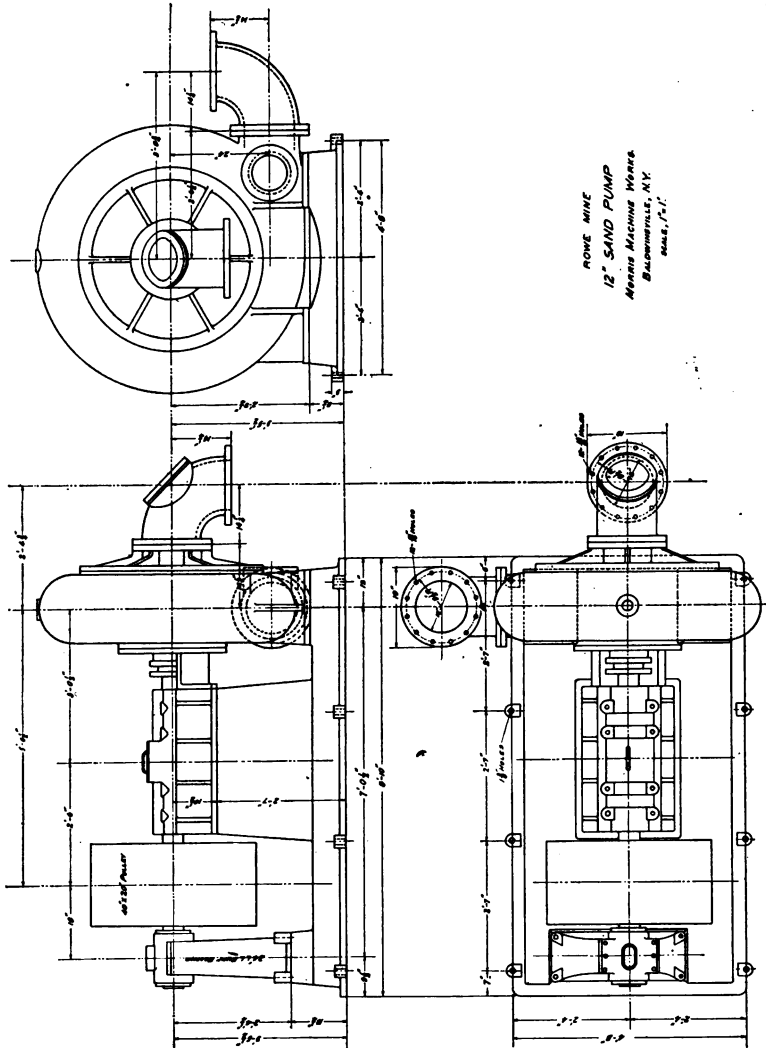


HYDRAULIC METHOD OF STRIPPING OVERBURDEN

er type of pump was used on the clear water or pressure line but it was soon abandoned, for the reason that the work of the giant was irregular, requiring frequent stopping. This could not be accomplished in the case of the plunger pump without shutting down the pump. The pump was located at some considerable distance from the giant and in practice it was found that telephonic communication was inadequate in the smooth running of the pumping apparatus.

On replacing the plunger pump with the two-stage centrifugal pump, shown in Plate 1, it was possible to get a pressure at the nozzle equal to that obtained with the plunger type and also a more steady stream of water with the advantage that the giant could be shut off partially or totally without materially increasing the pressure in the line. When

with iron ore and layers of sand carrying considerable nests of boulders. This overburden at times was excessively sticky and tenacious. Steam shovels handled it with difficulty when the clay layers were encountered.



away the returning stream of water gradually carried less and less of a load of material down to the lake. The sluicing was then abandoned and the hydraulic method of stripping installed.

The summary of operations for the Rowe mine are fairly shown in Plate 5 for the first five months work.

The double plant consisted of:

(a) Two 10-in. two-stage centrifugal pumps, for clear water. Each pump was directly connected to a 200-h.p. Allis-Chalmers motor. The details of this pump are shown in Plate 1. The pump was furnished by the Epping-Carpenter Pump Co., Pittsburgh, Pa., and cost \$2,625.00 f. o. b. Pittsburgh.

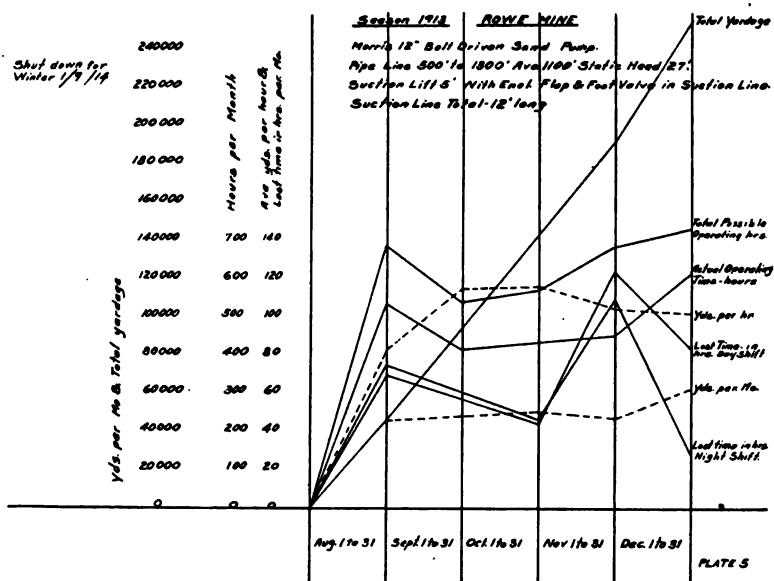
(b) Two sand pumps made by the Morris Pump Co. These sand pumps are of the centrifugal type with a 12-in. suction and a 12-in. discharge. Each pump was belt connected to a 250-h.p. Allis-Chalmers motor, 2,300 volts, 60 cycle, 3-phase, 7-speed. These pumps cost approximately \$1,000.00 each, f. o. b. Baldwinsville, N. Y.

The discharge pipe extended to a maximum of 2,400 ft. and was provided with gate valves so as to produce an artificial head. Each sand pump lifted 3,500 gal. per min. of which approximately 10 per cent. was sand. The pipe was 12-in. in diameter, spiral riveted, number 16 gauge steel, made by the American Spiral Pipe Company. The total cost of this pipe for both sand and clear water was \$2,000.00.

Details of the type of platform, etc., used at the Rowe mine are shown in Plate 3. This drawing illustrates the plan used at the Hillcrest mine which has been somewhat modified from the original designed at the Rowe. The pipe supporting the platform used at the Rowe was $4\frac{1}{2}$ in. in diameter as against 6 in. at the Hillcrest and these pipes were placed 16 ft. center to center at the Rowe and 10- by 16-ft. at the Hillcrest. The platform was subsequently replaced at the Hillcrest by a flat car bottom.

Plate 2 shows the details of the sand pump used at the Rowe mine. This sand pump was provided with a variable speed motor, belt connected to the pump, while at the Hillcrest the direct connected type of motor and pump is in service. In some cases this latter arrangement might not give enough speed variations for the different materials to be handled; and, also, the thrust of the pump is liable to cause hot journal boxes on the motor. Probably a better mechanical

Both the pressure and the sand discharge lines are laid on fairly regular grades and curves. Reference to Plate 5, where the topography is shown, will illustrate this. The sand discharge line is equipped with bolted joints and can be given a considerable curve both vertically and horizontally. This curving increases the friction head and causes heavy wear on the pipe where the bends occur. As usual, check valves are placed on both pipe lines where the pipe bends over into the pit to admit air when the pump is closed down and to permit the draining of the lines into the pit. The pressure at the nozzle is 70 lbs. per square inch.



A total of 1,500,000 cubic yards was moved hydraulically at the Rowe mine at an average cost of 6.7 cents per cubic yard. This cost covers labor, supplies, upkeep, and office expenses.

The labor necessary consisted of one motorman, one suction tender, one nozzleman, and two laborers. The nozzleman was paid 35 cents per hour; the others 30 cents per hour. The power necessary, which was 450-h.p., was paid for at the rate of 1¼ cents per kilowatt hour. The cost of labor (about one-half that of power) plus the cost of power, allowing for a reasonable repair item, was 4 cents per cubic yard.

Details of the performance of pumps, 1 and 2, are well shown in their record for October, 1914.

PERFORMANCE CARD NO. 1 PUMP.

	Day.	Night.	Total.
Actual hours worked by pump.....	285.25	360.75	646.00
Hours idle	38.75	11.25	50.00
Possible hours	696.00		
Day shift run, hours	285.25	41.0 per cent.	
Night shift run, hours	360.75	51.8 "	
Day shift lost	38.75	5.6 "	
Night shift lost	11.25	1.6 "	
Yards moved in month.....	40,000		
Yards moved per hour	61.9		

PERFORMANCE CARD NO. 2 PUMP.

	Day.	Night.	Total.
Actual hours worked by pump.....	249.75	342.00	591.75
Hours idle	74.25	18.00	92.25
Possible hours	684.00		
Day shift run, hours	249.75	36.5 per cent.	
Night shift run, hours	342.00	50.0 "	
Day shift, lost hours	74.25	10.9 "	
Night shift, lost hours	18.00	2.6 "	
Yards moved in month	75,000		
Yards moved per hour	126.7		

The best performance was that of pump number 2 in June, 1914, as follows:

	Day.	Night.	Total.
Actual hours worked by pump	215	292	497
Hours idle	97	78	175
Cubic yards moved	102,000		
Possible hours	672		
Time lost	175		
Day shift run, hours	215	32.0 per cent.	
Night shift run, hours	282	42.0 "	
Day shift, lost hours	97	14.4 "	
Night shift, lost hours	78	11.6 "	
Yards moved per hour	205.2		

Operations were begun at the Hillcrest mine on the 22nd day of April, 1915. Between that date and May 1st the work was principally devoted to getting the pumps started and experimenting with various devices; 11,127 cubic yards of material was moved during that time. The operation is planned to remove 1,000,000 cubic yards by hydraulic stripping. It is yet too early to arrive at a cost statement, but conditions and equipment being similar to those at the Rowe the writer is of the opinion that the cost will be nearly identical.

The following summary of the operations from May 1st to August 1st is complete and of great interest:

SUMMARY OF OPERATIONS TO JUNE 1, 1915.

	May 5 to June 1.	Total to June 1.
*Yardage moved	77,704 cu. yds.	96,127 cu. yds.
Number of hours, day shift....	201 hrs. 17 min.	253 hrs. 24 min.
Number of hours, night shift....	241 hrs. 37 min.	298 hrs. 57 min.
Total working hours	442 hrs. 54 min.	552 hrs. 21 min.
Cubic yards per hour.....	175 cu. yds.	174 cu. yds.
Total possible hours.....	532 hrs. 00 min.	
Average hours per shift.....	9 hrs. 51 min.	
Average cu. yds. per shift.....	1,728 cu. yds.	1,502 cu. yds.
Amount of water delivered.....	95,832,00 gals.	126,312,000 gals.
Percentage of solids.....	16.4 per cent.	15.4 per cent.

CAUSES OF SHUTDOWNS.

Moving pipe line.....	12 hrs. 47 min.
Repairing pump	29 hrs. 20 min.
Hot thrust bearing.....	36 hrs. 29 min.
Packing pump	2 hrs. 05 min.
Inspection	2 hrs. 20 min.
No power	4 hrs. 30 min.
Changing runner	1 hr. 00 min.

Total88 hrs. 31 min.

*7,296 cu. yds. moved May 1st, to May 5th; making a total of 85,000 cu. yds. for May.

SUMMARY OF OPERATION TO JULY 1, 1915.

	June 1 to July 1.	Total to July 1.
Yardage moved	59,728 cu. yds.	155,254 cu. yds.
Number of hours, day shift....	222 hrs. 15 min.	476 hrs. 44 min.
Number of hours, night shift....	275 hrs. 30 min.	573 hrs. 12 min.
Total working hours.....	497 hrs. 45 min.	1,049 hrs. 56 min.
Cubic yards per hour.....	120 cu. yds.	148 cu. yds.
Total possible hours.....	648 hrs. 00 min.	
Average hours per shift.....	9 hrs. 12 min.	
Average cu. yds. per shift.....	1,106 cu. yds.	
Amount of water delivered.....	81,833,900 gals.	208,145,000 gals.
Percentage of solids.....	14.8 per cent.	

CAUSES OF SHUTDOWNS.

Work on pipe line.....	54 hrs. 25 min.
Lowering scow	16 hrs. 05 min.
Repairing pump	72 hrs. 40 min.
Hot thrust bearing.....	3 hrs. 25 min.
Miscellaneous stops	3 hrs 40 min.

Total150 hrs. 15 min.

SUMMARY OF OPERATIONS FROM JULY 1 TO AUGUST 1, 1915.

	July 1 to Aug. 1.	Total to Aug. 1.
Yardage moved	63,195 cu. yds.	223,449 cu. yds.
Number of hours, day shift....	263 hrs. 50 min.	740 hrs. 34 min.
Number of hours, night shift....	286 hrs. 40 min.	859 hrs. 52 min.
Total working hours	550 hrs. 30 min.	1,600 hrs. 26 min.
Cubic yards per hour.....	124 cu. yds.	138 cu. yds.
Total possible hours.....	696 hrs. 00 min.	
Average hours per shift.....	10 hrs. 21 min.	
Average cu. yds. per shift.....	1,275 cu. yds.	
Amount of water delivered.....	107,017,200 gals.	315,163,100 gals.
*Percentage of solids.....	13.0 per cent.	

CAUSES OF SHUTDOWNS.

Work on pipe line.....	57 hrs. 15 min.
Repairing pump	30 hrs. 35 min.
Hot thrust bearing.....	30 min.
Waiting for and setting up pump.	66 hrs. 05 min.
Miscellaneous stops	1 hr. 30 min.

Total145 hrs. 30 min.

*Allowance made for 200 gals. per min. seepage into the pit.

SUMMARY OF OPERATION FROM AUGUST 1 TO SEPTEMBER 1,
1915.

	Aug. 1 to Sept. 1.	Total to Sept. 1.
Yardage moved	95,539 cu. yds.	319,589 cu. yds.
Number of hours, day shift....	294 hrs. 10 min.	1,034 hrs. 44 min.
Number of hours, night shift...	345 hrs. 40 min.	1,205 hrs. 32 min.
Total working hours.....	639 hrs. 50 min.	2,240 hrs. 16 min.
Cubic yards per hour.....	149 cu. yds.	142 cu. yds.
Total possible hours.....	720 hrs. 00 min.	
Average hours per shift.....	10 hrs. 39 min.	
Average cu. yds. per shift.....	1,592 cu. yds.	
Amount of water delivered.....	137,437,600 gals.	452,700,700
Percentage of solids.....	13.2 per cent.	

CAUSES OF SHUTDOWNS.

Work on pipe line.....	50 hrs. 15 min.
Repairing pump	11 hrs. 00 min.
Setting up new pump.....	10 hrs. 50 min.
No power	8 hrs. 05 min.

*Allowance made for 200 gals. per min. seepage into the pit.

It is interesting by way of comparison to know that in the pebble phosphate district of Florida the hydraulic method of removing the overburden and also of removing the pebble phosphate is used exclusively, some of the larger companies employing as many as 25 of these different dredging units at one time. Up to the present time it has been the custom to use 10-in. pumps in the phosphate fields for this work but some 12-in. pumps are now being installed. The depth of the overburden is shallow in the phosphate region compared to that in the iron mines, the average being about 20 ft., and as the bed of phosphate is also shallow the sumps into which the materials are washed by the hydraulic giant have to be moved much more frequently than in the iron mines. For this reason in the phosphate mines it is customary to use long suction lines on the pump. A suction hose is placed next to the suction disc on the pump and through the flexibility gained by this suction hose, and with a suction line that is gradually increasing until about 200- to 250-ft. is reached, considerable area can be covered with the one setting of the pump and

several different sumps reached with this long suction line. It is customary to also carry a much higher pressure on the giant nozzle than that used in the iron mines. Practically all the pumps in the phosphate region used in supplying water to the giants are designed for a pressure of 175-lbs. at the pump which results in a pressure of from 150- to 170-lbs. at the giant nozzle. The average output of the 10-in. pump in the phosphate district is 2,000 yards per day of 24 hrs. with a probable actual operating time of about 20 hours.

The economic limits for hydraulic stripping are very sharply defined and the work can easily be carried to a point which ultimately necessitates too much hand work. Experience at the Rowe mine has shown that it is advisable to leave 6- to 8-ft. of surface on top of the orebody to be cleaned up later by the steam shovel.

Due to the heavy repair work on the sand pump it is necessary to keep the pump and pump line free from boulders, brush, roots, etc. This is best done by hand picking, the material accumulated being later removed by the steam shovel at the clean up.

Stripping hydraulically on the Cuyuna range has been a marked economic success as compared with steam shovel stripping under similar conditions. The rapidity of such operations especially recommends the method.

Acknowledgment for valuable assistance in the preparation of this paper is due Mr. J. C. Barr, General Manager of the Rowe mine; Mr. Frank Hutchinson, Chief Engineer of the Rowe mine; Mr. Wilbur Van Evera, Superintendent of the Hillcrest mine, and Mr. P. J. McAuliffe of the Morris Pump Company. The writer takes this opportunity to thank these gentlemen for their many courtesies.

DRAG-LINE STRIPPING AND MINING, BALKAN MINE, ALPHA, MICH., MENOMINEE RANGE, MASTODON DISTRICT.

BY CHARLES E. LAWRENCE, PALATKA, MICH.*

The Mastodon district of the Menominee range is located five miles southwest from Crystal Falls, on Section 12/13, 42-33. Ore shipments began in 1882 and continued for several years, until over 430,000 tons had been shipped, after which the district was abandoned.

Five years ago the E. J. Longyear Company, of Minneapolis, then exploring in Iron County, secured options to explore on adjoining lands and in three years of continuous drilling located the present Balkan mine and other ore deposits. One of these deposits was sold to Pickands, Mather & Company, of Cleveland, Ohio, and, in the spring of 1913, this company began stripping the overlying surface. The ore outlined by drilling was covered by a cedar-tamarack swamp, through which a small stream flowed to Buck Lake, one mile distant. A new channel was dug for this stream, far enough away to carry its waters safely past the open pit. Next a shaft was sunk, 150 feet north of the orebody, in the slate footwall to serve, first, to drain the swamp of water, and, later, for permanent mining. The sinking of this drop shaft was hindered by a stratum each of quicksand, heavy blue clay, cement hardpan, and boulder gravel, together with a large amount of water, before the slate ledge was encountered at a depth of 56 feet.

At 80 feet depth, a water sump was made and pumps were installed to care for the water coming into the shaft. The shaft was sunk to a depth of 132 feet, or to the 1st level drift. This drift was driven south 500 feet in slate and ore, and raises put up to the sand to tap the water. These raises failed to accomplish this, however, because of impervious cement hardpan and clay which held the water above them, so

*General Superintendent, Pickands, Mather & Co., Menominee Range.

several 5-in. pipes were driven to this drift to draw off the water from the pit. A valve was placed at the bottom of these pipes so that the flow of water could be stopped in case of accident to the mine pumps. These pipes were driven in short lengths, so that pieces could be uncoupled as the stripping progressed.

Drag-Line Stripping—The area stripped is an oval with

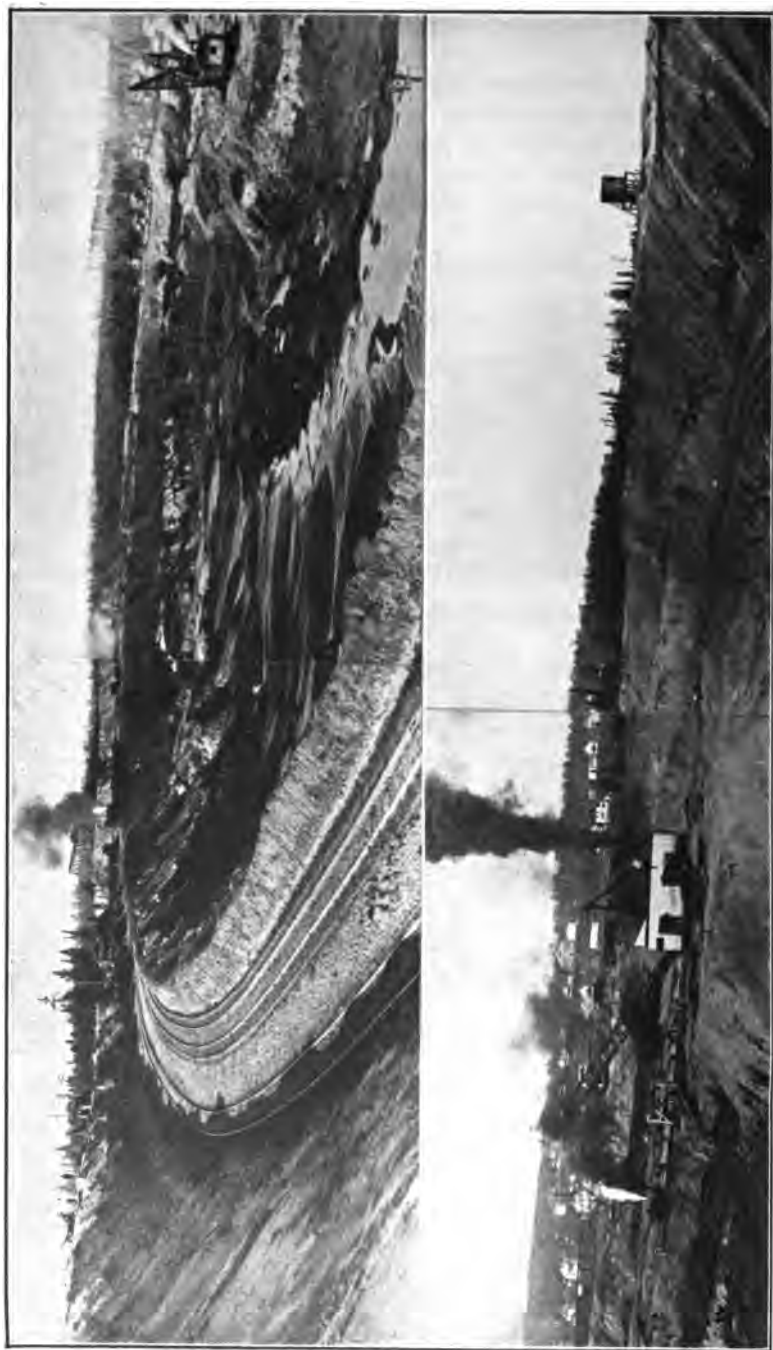


BALKAN MINE—STARTING DRAGS IN SWAMP, MAY, 1914



BALKAN MINE—JULY 1, 1914. VIEW LOOKING SOUTHEAST, SHOWING SOUTH HALF OF CUT.

the long axis SE-NW and approximately 1150 feet by 900 feet at the surface. The slope angles are two to one in the fine sand on top and run to one to one in the underlying gravel and clay; a steeper slope could have been maintained if the material had been dry. The track incline of the pit is a spiral on a 2.6 per cent. grade and extends to ore at a depth of 85 feet from the original surface. The depth of stripping varies



VIEW OF BALKAN PIT FROM EAST SIDE, LOOKING WEST TO NORTHEAST

from 60 feet on the northwest side to 108 feet on southeast side.

In May, 1914, a contract was begun with the Winston Brothers Company, of Minneapolis, to remove the surface, consisting of 1,200,000 cubic yards. This company used two drag-line excavators, one of the Marion type and the other of the Bucyrus type. These excavators each have an 85-foot boom, a 4-yard dipper and a 24-foot turntable. They are mounted on hardwood rollers running on 4-inch plank. When the machine is in operation, angle irons are placed on both sides of the rollers. If the machine is to be moved, the irons are removed and the machine pulls itself along with its own power. The working weight of the machines is approximately 300,000 pounds.

The principal factors leading to the selection of this type of machine, instead of the familiar steam shovel, were the large quantity of water in the surface and the texture of the material. The fine sand and water made a bottom which would not support a steam shovel; a drag-line machine, however, remains at the surface and makes its cut below its own elevation. The limit of depth depends upon the slope taken by the material in question, in this case approximately 30 feet.

The material removed is handled through a hopper into 4-yard western dump cars, ten to a train, and hauled to the dump, one-half mile distant, by 15-ton locomotives at an average rate of 2,000 yards per 10-hour machine shift.

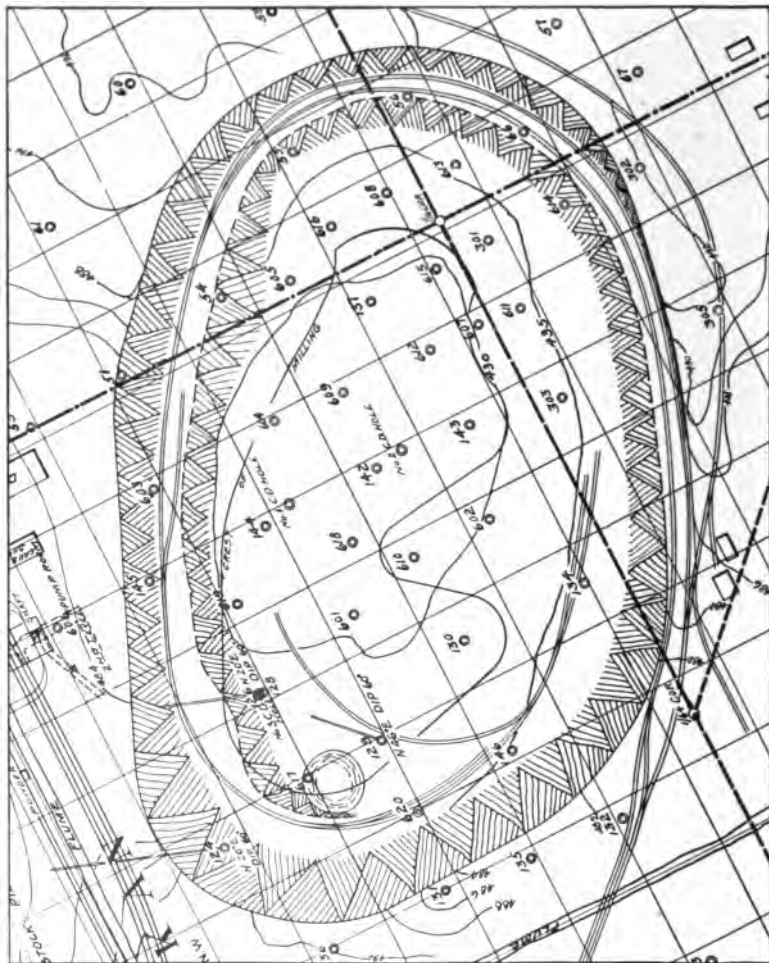
Under the terms of the contract, the mining company handled the water in the pit, employing motor-driven Morris 6-inch centrifugal pumps with 8-inch spiral discharge pipes. During the 1914 season the water pump averaged 1,200 gallons per minute, but the amount has gradually slacked off to 500 gallons at the present.

When the stripping had reached a depth of approximately 60 feet, some of the clay banks began to cave and slide off. These banks were dressed with evergreen boughs, or gravel from the pit was dumped over them. This gravel contains a large percentage of clay and has baked solidly in place, effectively stopping the caving.

As soon as ore was cleaned up, the mining company cribbed the banks closely and filled the crib with gravel. Next to this cribbing they have dug a ditch in the ore completely around the orebody, so that all water made in the pit is con-

ducted to one sump. A 20-foot berm of ore is left to maintain the ditch and cribbing holding up the sides of the pit.

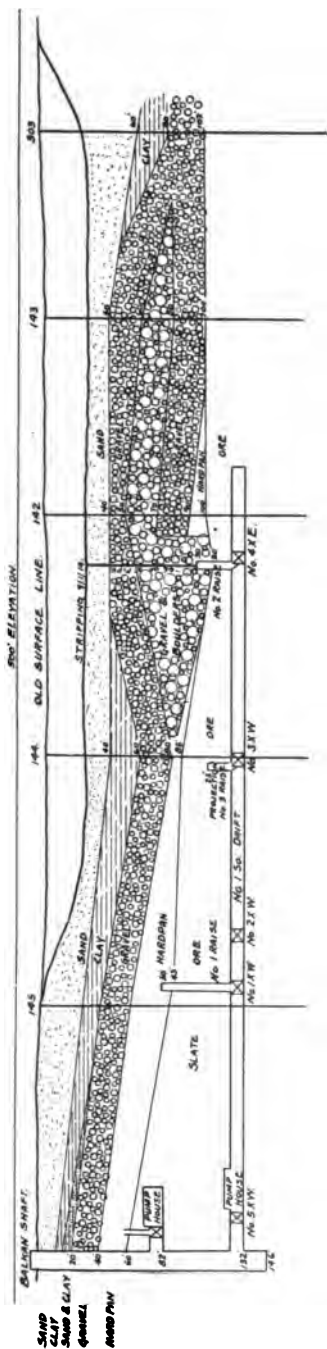
To shake the ore, the contractors are drilling holes with two gasoline churn drills and five steam-piston drills on tri-



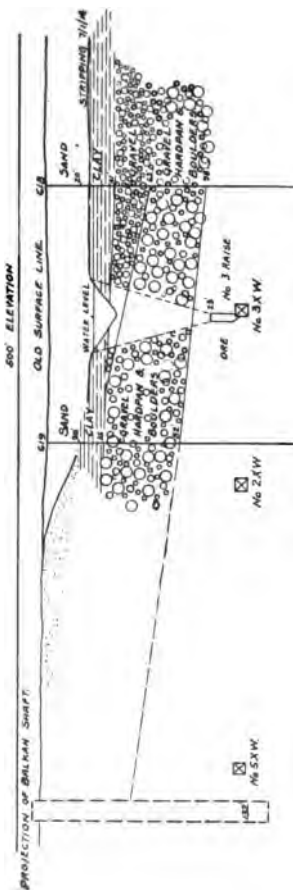
MAP SHOWING AREA OF STRIPPING, BALKAN MINE, CRYSTAL FALLS, MICH. JAN. 1, 1915

pod. The gasoline drills are putting down vertical holes from 20 to 30 feet deep and about this same distance apart. The piston drills are placing 15-foot holes between the rows of deep

CROSS SECTION X:



CROSS SECTION Y:



CROSS-SECTIONS, BALKAN MINE. CRYSTAL FALLS, MICH. JULY 27, 1914.

holes. The dynamite used is 40 per cent. glycerine and it is fired with a battery.

The stripping contract was completed about the middle of August, and the drag-line machines at once started removing ore to a stockpile. Here the ore is mixed to maintain a uniform grade and the chunks are sledged. Ore is being put into stock at a rate of 5000 tons per day and a total of 200,000 tons will be removed from the pit this season, in three months of work. The trestle for the stocking of ore is 600 feet long and 251 high and so constructed that a train and engine can be run its full length. This permits of spreading the ore and consequently facilitates the grading and sledging of it. Steam shovels will re-load the ore in stock for shipment to Escanaba.

Thus the drag-line machines, working under severe conditions, proved successful for handling the mushy ground occurring in swamps. Likewise they proved successful when used for mining ore of a medium hard nature at the same property, for the first time in the Lake Superior district.



PENNINGTON MINE, PENNINGTON MINING Co., CROSBY, MINN.



KENNEDY MINE, ROGERS-BROWN ORE Co., CUYUNA, MINN.



CROSBY, MINN.—TYPICAL MINERS' HOMES



**HYDRAULIC METHOD OF STRIPPING OVERBURDEN AT THE
HILLCREST MINE, CUYUNA RANGE**



CROFT MINE, MERRIMAC MINING CO., CROSBY, MINN.



ARMOUR NO. 2, INLAND STEEL CO., CROSBY, MINN.



STEVENSON MINE, CORRIGAN, MCKINNEY & CO., MESABI RANGE, 1913



THOMPSON MINE, INLAND STEEL CO., CROSBY, MINN.

SECOND ANNUAL FIRST-AID CONTEST.

BY EDWIN HIGGINS, PITTSBURGH, PA.*

The second annual first-aid contest, in connection with the meeting of the Lake Superior Mining Institute, was held September 6, 1915, at Ironwood, Mich., in the presence of about 1800 spectators. The baseball park on the eastern edge of the town, in which the contest was held, afforded ample seating capacity for the Institute members and their friends. Many automobiles were parked in the field. The field places for the contesting teams were arranged in the form of an arc of a circle, in front of the grandstand and bleachers. The contest was carried out under the auspices of the Gogebic Range Mining Association. Fourteen teams took part in the contest; the following qualified for prizes, finishing in the order indicated:

First, Verona Mining Company, Menominee Range; second, Oliver Iron Mining Company, Mesabi Range; third, Odanah Iron Company, Gogebic Range; fourth, Montreal Mining Company, Gogebic Range; fifth, Judson Mining Company, Menominee Range; sixth, Newport Mining Company, Gogebic Range. The one-man event was won by the Oliver Iron Mining Company, Mesabi Range. The three-men event was won by the Republic Iron & Steel Company, Cambria mine, Marquette Range.

PRELIMINARY WORK IN ARRANGING CONTEST.

It has been suggested that it would be desirable to set forth in detail the various steps in arranging and carrying out this contest, as such information may be of value as a matter of record. Moreover, it will afford an opportunity for constructive criticism, with a view to improving the meets from year to year.

A committee of eight, on arrangements, was first appointed by the Gogebic Range Mining Association. On June

*Engineer, U. S. Bureau of Mines.

10, 1915, the following letter, and rules governing the contest, (with corrections conforming to subsequent changes) were sent to all mining companies of the Lake Superior region:

"To Mine Operators of the Lake Superior District:

"One of the attractions of the forthcoming meeting of the Lake Superior Mining Institute at Ironwood, Mich., will be a first-aid contest. The date of the Institute meeting has not yet been announced. It may be stated, however, that it will be about the end of August. The exact date, with other details, will be sent to you at an early date.

"You are assured that the first-aid contest will be conducted in a manner least calculated to meet the disapproval of participants and spectators. Attractive and adequate prizes



BLEACHERS AND CONTESTANTS

will be offered to the contending teams. As far as possible, officials will be made up of disinterested parties. Judges will be secured from outside the Lake Superior district. Following a plan that has met with much success in meetings held elsewhere, and one that provides a fairer test of a knowledge of first aid work, the events for the contest will not be announced until the teams are on the field ready for work. It may be said, however, that there will be no catch problems given and that the work will be confined to such accidents as commonly occur in and about metal mines. The meet is open to all mining companies in the Lake Superior district.

"Contests of this kind are recognized stimulants of first-aid work, the value of which has been so completely dem-

onstrated, and it is hoped that this meet will be given the hearty support of the mining companies of the district.

"The meet will be conducted under the auspices of the Gogebic Range Mining Association. Entries for the contest must be sent to Mr. L. C. Bishop, Secretary, Ironwood, Mich., prior to July 15th. There is appended hereto a list of rules by which the contest will be governed."

RULES GOVERNING FIRST-AID CONTEST.

1. A team is composed of six men, one of whom shall be captain. Any employe of a mining company, excepting physicians or trained nurses, may be a member of a contesting team.

2. The captain shall elect a patient and designate the member or members of the team to perform an event. The patient must be clad in tights.

3. Members of teams, except the patient, will wear the following described uniform: Hat or cap, coat, trousers and shoes, all of white duck.

4. The captain will control his team in their field work by giving audible commands.

5. The captain may select himself as one of the members who will perform the event. The captain may contest in team events.

6. The captain or other members of a team will not prompt the person performing the event unless he is one of the performers. This does not apply to full team events.

7. At the conclusion of an event, the captain will raise his right hand and announce his team number. The team will remain at post until relieved by the judge.

8. Teams will bring their own first-aid material, including bandages, splints, blankets, stretchers, etc. Members will not be allowed to leave the patient to secure material.

9. The triangular bandage will be the standard used in this contest, but equal credit will be given for the proper use of the roller bandage.

10. All splints must be fully prepared on the field for each event requiring their use. Specially designed splints may be used, but they must be assembled during the time of each event requiring their use.

11. No practicing will be allowed on the field before the contest. Teams must do the work called for by the event, and
NO MORE.

12. No event shall be started with bandages already folded.
 13. The teams will be numbered consecutively and will occupy consecutive positions on the field.

14. The judges will perform their work progressively, judging as many teams in each event as may be determined and announced before the contest starts.

15. In events involving resuscitation, rescue of patient, and stretcher drill, the judges may require the teams to perform separately. Only manual artificial respiration shall be used.

16. Each judge will record the team number, event, and discount for each team judged, sign his name and deliver the same to the recorder.

17. The recorder will foot up the discounts and mark points made by each team in each event. The total points will be divided by the number of events and the quotient will be the average for each team for the entire contest.

18. Time will not be an element unless the team or men performing the event run over the allotted time, or fail to give treatment properly. Failure to finish an event in the allotted time shall be discounted one point for each minute additional required. All events start and stop with the sounding of a gong.

19. In the event of ties for first place, the teams tying will re-contest for first, second and third place; in the event of ties for second place, the teams tying will re-contest for second and third place; and in the event of ties for third place, the teams tying will re-contest for third place.

20. The following discounts, which have been adopted by the American Mine Safety Association, will apply:

Not doing the most important thing first	6
Failure of captain to command properly	1
Slowness in work and lack of attention	2
Failure to entirely cover the wound or ignorance of location of injury	4
Ineffective artificial respiration	11
Splints improperly padded or applied	6
Tight, loose, or improperly applied bandages	5
Insecure or "granny" knot	4
Unclean first-aid material	3
Failure to have on hand sufficient and proper material to complete a dressing	3
Lack of neatness	2
Awkward handling of patient	4
Assistance lent by patient	3
Tourniquet improperly applied	7
Failure to stop bleeding	8

Not treating shock	5
Failure to be aseptic	7
Improper treatment	12
Failure to temporarily control hemorrhage previous to application of tourniquet	7

About three weeks before the date set for the contest, at a meeting of the arrangements committee, various sub-committees were appointed to take care of all details that could be foreseen. In order that the positions of the teams on the field might be printed on a program for the benefit of the spectators, lots were drawn for the assignment of positions. A letter was then addressed to the contesting teams advising them of their position on the field, requesting that they send in the names of the team members, and giving them instructions as to arrangements that had been made for them at Ironwood.

It was recognized that the selection of judges for the contest, and the formulation of rules, were two important factors. Insofar as possible, without departing from practice in the Lake Superior region, the general rules governing the National First-Aid Contest at San Francisco, were adopted. This contest was held September 23, 1915, and the rules governing it were approved by a committee made up of representatives of the American Red Cross, American Mine Safety Association and the Bureau of Mines.

The services of seven judges, all except one of whom reside in Terre Haute, Ind., were secured for the contest. These gentlemen are all physicians and their selection was approved by the American Red Cross. They came to Ironwood as guests of the Institute and the Range Association.

As to the selection of the problems for the contest, inasmuch as these were to be kept secret until announced on the field, the arrangements committee decided to have the writer select a list of problems based upon accidents which commonly occur in the Lake Superior region and send them to the chairman of the judges, who would select and keep secret those to be given at the contest. This arrangement was followed out to the letter. It was not until the last few hours of the contest that even the judges knew of the events that had been selected. The chairman of the judges thought it proper, and wisely too, to go over the problems carefully with his colleagues and arrive at a definite and uniform basis for judging the events.

THE CONTEST.

At 9:20 a. m., September 6, the 14 contesting teams were

formed in a double line in the parking of the Chicago & Northwestern railroad station. At 9:30 o'clock, escorted by the Newport and Norrie bands, and the officials and judges of the contest, the teams started for the baseball park. The parade was about two blocks in length and the team members presented an attractive appearance in their uniforms of white duck. At 9:50 o'clock the teams entered the field and took the places assigned to them. After instructions had been given to the contestants by the chairman of the judges, the teams made ready to receive the first problem. There were seven judges in all, each of whom had two teams to look after. On the sounding of one bell, the problem, typewritten on a sheet of paper, was delivered to the teams. At the same time



GRAND STAND AND CONTESTANTS

the problem was announced to the spectators. The contestants were allowed two minutes to study the problem, at the end of which time two bells were rung, signalling the start of the event. On the ringing of three bells the time limit set for the problem expired.

The six problems selected for the contest are submitted herewith. Owing to a lack of time, however, only problems number 1, 2, 3 and 5 were worked out.

Problem No. 1—One-man event. Miner found a distance of 20 feet in closely caved workings with palm of right hand lacerated and bleeding profusely; in shock. Treat and transport 20 feet to more open workings, carry 20 feet by shoulder lift. Time 15 minutes.

Problem No. 2—Three-men event. Miner found in caved

drift ($2\frac{1}{2}$ to 3 feet high) in bad air after explosion of powder, unconscious from gases, burns of face and hands. Transport 20 feet and administer artificial respiration by Schaefer method for one minute. Treat burns. During transportation one of the rescuers is overcome, becoming unconscious; rescue and administer proper treatment. Time 15 minutes.

Problem No. 3—Team event. Miner caught by fall of rock; left ear torn off; left shoulder dislocated; left thigh broken in upper third of leg (compound fracture) with profuse bleeding; in shock. Treat and transport 30 feet on improvised stretcher. Time 20 minutes.

Problem No. 4—Team event. As a result of fall motor-man found lying face down on track, live wire extending across right cheek and right hand, unconscious, right hand and right side of face badly burned. In shock. Administer proper treatment. Time 15 minutes.

Problem No. 5—Team event. Miner walks into blast; face badly lacerated over left cheek bone with profuse bleeding; severe laceration of upper part of abdomen; back of right hand lacerated, blood oozing; fracture of right knee cap; unconscious from powder gases; treat on spot (gas has cleared away), transport 40 feet. Time, 20 minutes.

Problem No. 6—Team event. Miner found under fall of rock, lying on side with back broken in lumbar region. Treat injury and shock, transport 20 feet on improvised stretcher. Time, 10 minutes.

During the contest the spectators were favored with selections by the Newport band. With the exception of a slight delay in starting for the field, occasioned by the late arrival of one of the teams, the contest was carried out with a minimum of confusion and untoward incidents, and in the best of spirit. The most striking feature of the contest was the uniformity in the costumes worn by the contestants. This feature was especially impressed on those of the spectators who had seen such contests in various parts of the country, where it is the usual custom for the teams to appear on the field in a variety of costumes, and often in their street clothes.

At the close of the contest the averages were figured out and the standing of the various teams announced. A feature of the closing scenes on the field was the presentation to the winning team of the American Red Cross medals. The chairman of the judges made the presentation speech and the med-

als were pinned upon the breasts of the team members by five ladies.

Following is a list of the teams that took part in the contest, with their field positions:

- No. 1—Oliver Iron Mining Company, Mesabi Range.
- No. 2—The Montreal Mining Company, Gogebic Range.
- No. 3—Odanah Mining Company, Gogebic Range.
- No. 4—Verona Mining Company, Menominee Range.
- No. 5—Oliver Iron Mining Company, Gogebic Range.
- No. 6—Judson Mining Company, Menominee Range.
- No. 7—The Castile Mining Company, Gogebic Range.
- No. 8—Colby Iron Mining Company, Gogebic Range.
- No. 9—Republic Iron & Steel Company, Mesabi Range.
- No. 10—Newport Mining Company, Gogebic Range.
- No. 11—Cleveland-Cliffs Iron Company (Ishpeming-Republic).
- No. 12—Pickands, Mather & Co., Mesabi Range.
- No. 13—Republic Iron & Steel Co., Marquette Range.
- No. 14—Cleveland-Cliffs Iron Company (Negaunee-Gwinn).

PRIZES.

The prizes, which were awarded (with the exception of the medals) at the Ironwood Commercial Club in the evening, were as follows:

First prize, \$175 in cash to defray the expenses of the team from Ironwood to the State Fair at Minneapolis and return, in company with the members of the Institute; donated by the Gogebic Range Mining Association, awarded to the team of the Verona Mining Company, Menominee Range. This team also received the bronze medals awarded by the American Red Cross.

Second prize, \$50 in cash, donated by the E. I. duPont deNemours Powder Company, awarded to the team of the Oliver Iron Mining Company, Mesabi Range.

Third prize, \$30 in cash, donated by the Lake Superior Mining Institute, awarded the team of the Odanah Iron Company, Gogebic Range.

Fourth prize, \$20 in cash, donated by the Lake Superior Mining Institute, awarded to the team of the Montreal Mining Company, Gogebic Range.

Fifth prize, two thermos bottles, a razor, a pair of trousers and two suits of underwear, awarded to the team of the Judson Mining Company, Menominee Range.

Sixth prize, 50 cigars, a rocking chair, a carpet sweeper and five pounds of butter, awarded to the team of the Newport Mining Company, Gogebic Range.

The one-man event was won by a performer from the team of the Oliver Iron Mining Company, Mesabi Range. The prizes were a kodak to the performer and a cigar case to the patient. The three-men event was won by contestants from the Republic Iron & Steel Company, Cambria mine team, Marquette Range, and consisted of a Stetson hat, solid gold cuff links, and a safety razor to the performers, and a pipe to the patient.

The articles of merchandise included in the above prizes were donated by the following firms of Ironwood: Jussen



SPECTATORS IN AUTOMOBILES

& Trier, M. F. McCabe & Co., Davis & Fehr, J. P. Bekola, Wm. D. Triplett, Gamble & Mrofczak, C. M. Bean, L. Ladin, Anderson & Oksa, Mullen Bros., Ironwood Pharmacy, City Drug Store, C. E. Erickson Hdw. Store, W. Ekquist, Buss Creamery.

COMMITTEES, OFFICIALS, AND JUDGES OF THE CONTEST.

Following are the various committees, the names of the officials, and the judges who served in the contest:

Committee on Arrangements—P. S. Williams, Chairman, Edwin Higgins, L. C. Bishop, B. Brockbank, A. E. Redner, John Mildren, A. A. Bawden, B. D. Shove.

Committee on Grounds—L. C. Bishop, H. W. Byrne.

Committee on Awards—P. S. Williams, L. C. Bishop, A. E. Redner.

Contest Director—Edwin Higgins, Bureau of Mines.

Recorders—Frank Blackwell, Oscar E. Olson.

Time-Keeper—O. M. Schaus.

Judges—Dr. August F. Knoefel Chairman (President American Mine Safety Association), Dr. M. R. Coombs, Dr. C. N. Coombs, Dr. A. M. Mitchell, Dr. Rudolph Duenweg, Dr. R. L. Woodard, all of Terre Haute, Ind., and Dr. G. D. Scott, of Sullivan, Ind.

CONCLUSION.

It appeared to be the general impression that the contest was a very successful one. Several of the judges stated that they had seen few contests carried on with less friction and confusion or in which the teams presented a better appearance on the field. Doctor Knoefel, in his address to the winning team, emphasized the fact that there were 14 excellent teams on the field, and that there was none that need be ashamed of the exhibition it made. That the teams were very evenly matched and that their work was highly efficient is born out by the fact that the percentage of eleven teams was above 84. In fact, there were only nine points difference between the first and eleventh team, the winning team having a score of 93¼%, and the team standing eleventh a score of 84¼%.

The writer feels that the operators of the Lake Superior mining region have every cause to be proud of the wonderful advance that has been made in first-aid work during the past two or three years. The exhibition given at Ironwood reflects credit on both the operators and the men behind the splints and bandages.

In order to make the work done in this contest of particular value to the teams, the committee on arrangements decided to send score cards to the various teams so that they might see the reasons why they were discounted and profit by the mistakes thus indicated.

The writer, on behalf of the Gogebic Range Mining Association, takes this opportunity of thanking the officials, judges, donors of prizes, and others who did much to make this contest a success. Thanks is also extended to the Newport Mining Company and to the Oliver Iron Mining Company for the donation of the services of the Newport and Norrie bands.

MATTERS OF INTEREST TO OPERATORS REGARDING THE CUYUNA DISTRICT.

BY CARL ZAPFFE.*

Although the development of the Cuyuna Iron Ore District is as yet scarcely begun, the operations at a few of the properties have already revealed certain features that stamp the district as with a trade mark. Some of these are merely interesting, but others are of great importance, and if they have not already given the district great publicity, they surely ought to. The purpose of this paper is to present and describe some of these features.

In preparing this paper the author has conferred with the following men of experience and prominence in the Cuyuna district: John S. Lutes, General Superintendent of the Tod-Stambaugh Company operations; Wilbur Van Evera, in charge of the Hillcrest mine of the Hill Mines Company; D. C. Peacock, consulting engineer, Brainerd, Minnesota, and W. A. Barrows, Jr., metallurgist and consulting iron ore expert, Brainerd, Minnesota.

SURFACE CONDITIONS.

The overburden is a prevailing sandy glacial drift. Interbedded with the sand are layers of clay, hardpan or gravel. Boulders are scarce and are purely local when abundant, and thus far have not seriously interfered with or rendered costly any shaft or stripping operation. The operations up to the present indicate that the sinking of a shaft through the surface in the Cuyuna district is not difficult.

To date nine drop shafts and five lath shafts have been sunk and six pits have been opened. The wooden drop shaft at the Kennedy mine was the most difficult and troublesome of all to sink and the Adams concrete shaft was slow in dropping, but aside from these, so far as the surface itself is concerned, no serious troubles have arisen anywhere. The Ken-

*Geologist, Brainerd, Minn.

nedy shaft, sunk in clay territory, encountered considerable water and many runs of quicksand; the Barrows shaft, located in a sand belt, struck a little quicksand. The Brainerd-Cuyuna drop shaft, however, penetrated 25 ft. of quicksand without trouble, and all the other shafts had virtually no difficulties at all or no unusual ones.

Six of the nine drop shafts are of concrete. One of these penetrates 123 ft. of surface, another 105 ft., and the other four each about 63 feet. The 123 ft. overburden is the deepest penetrated by any shaft in the district. Three other wooden shafts are sunk in 100 ft. of surface. In the productive area the surface varies from 14 to 240 ft. in depth, but is prevailingly nearer 100 ft. deep, and in only two of all the operated properties is the surface over 100 ft. deep. Only one concrete shaft was sunk in very wet ground. Reviewing the situation, surface conditions did not make concrete shafts imperative.

The best all-around record for wooden drop shaft sinking was made at the Wilcox mine. It is located in new territory and was a mile and a half away from a railroad at the beginning of operations. A shaft measuring 6 by 16 ft. inside at the bottom was sunk through 66 ft. of sand and gravel and a 25-ft. layer of clay resting on the ledge, in the brief period of three months—an average of about one foot per working day of one, two or three shifts at different times. The Iron-ton lath shaft measures 6 by 14 ft. and was sunk through about 100 ft. of surface in 44 days. The first shaft ever sunk in the district was a lath shaft which went rapidly through 80 ft. of surface at the edge of a large muskeg swamp.

WATER.

The early predictions invariably were that much water would be encountered, but the actual experience in nearly every case has been just the reverse. Many of the properties that were confidently expected to produce large flows have agreeably surprised their operators. In several shafts the normal flow during sinking was about 100 gallons per minute. Once the shaft is in rock, water is generally never struck until the contact between ore and wall rock is reached, or even until the orebody itself is partly developed by drifting. The flow then increases by about 1,000 gallons. Even where the deposits are widely opened, the larger flows are only from 1,600 to 2,000 gallons. The largest flow that has ever been encount-

ered at any one time is about 3,000 gallons, and that was due, in some measure, to the very rapid opening of several lenses of ore. Nothing is now known that need cause any worrying over heavy pumping.

PIT OPERATIONS.

The earlier ore deposits found were all long and narrow, and therefore stripping was not considered; but in the course of time wider and deeper deposits were found and today there are six pits, other companies are about to begin stripping, and for still other properties it is surely feasible. Four of the six pits were dug entirely with steam shovels, one was dug largely by hydraulic methods and finished with steam shovels, and another is now following this latter plan. In the first pit ever dug, the Pennington, the total surface moved was 1,350,000 yards. The pit is about 1,000 ft. long and 600 ft. wide at the crest. The entire operation was cramped for room and involved heavy grades and switchbacks; nevertheless, in the remarkably short time of 180 days, working double shift, two shovels moved 1,250,000 yards,—a rate of 104,000 yards per month per shovel. In the following 87 days these shovels moved an additional 100,000 yards of dirt and shipped 100,000 tons of ore. The overburden was virtually all sand, a few small layers of interbedded clay helping to hold up the banks on a slope a little flatter than 1 to 1.

Recently the western half of the Armour No. 1 property has been stripped. This pit is about 600 ft. in diameter at the crest. This stripping was possible because it could be carried on in part through the adjoining Pennington pit, and under the following conditions—only part of the stripping to be dumped on a high dump, a sandy overburden, well-drained ground yet prevailing wet weather—one shovel, working double shift, in 148 days moved 772,000 yards—a rate of 154,000 yards per month. These are performances seldom equalled anywhere.

Hydraulic stripping is an innovation in Lake Superior mining. It was first undertaken at the Rowe mine, and is now also proving successful at the Hillcrest, both mines of the Cuyuna district. At the Rowe mine, after various experiments on a small scale, a 12-in. centrifugal pump belt-connected to a 250 h.p. motor, was found satisfactory and installed. This unit was set on a specially built platform mounted on six 4½-in. drill casings previously sunk to ledge. At the Hillcrest

mine the pump is directly connected to a 300 h.p. motor, and this unit is set on an old railroad flat-car as a platform mounted on six 6-in. drill casings. As the overburden is removed, from time to time this platform is lowered on these pipe-guides until the desired depth is reached. The centrifugal pump is set near the end of the platform and a 12-in. suction overhangs. The fresh water used is pumped from a lake nearby through a 12- or a 14-in. feed-pipe and is delivered to the pit through a 3½- or a 4-in. giant nozzle under a pressure of from 50 to 80 pounds. This pump, also electrically driven, is installed at a lake and delivers from 3,000 to 4,000 gallons per minute. The water is directed against the banks of the pit, and the material as it washes down from the banks flows over to the platform, where the centrifugal pump sucks up the dirt-laden waters. The discharge is from a quarter to one-half mile or so away on low ground.

This method works admirably in sandy or loamy soil. Ordinarily a property would not be entirely stripped by hydraulic methods, because clay cannot be moved advantageously. Stones larger than five inches in diameter cannot be easily sucked up and carried away in the discharge and must be moved by other methods. A steam shovel must invariably be used to finish the job, just as the wheelbarrow and hand shovel must follow and finish the job where a steam shovel is the prime mover. A unit such as now used is capable under favorable conditions, of moving about 4,000 yards of dirt in a 22-hour working day. The maximum yardage per month ever attained in this district was 102,000 yards, at the Rowe mine. At the Hillcrest mine the average per month for the first three months of operation has been 70,974 yards.

The essentials for a hydraulic operation in the Cuyuna district are as follows: (1) a sandy soil or overburden, (2) a convenient fresh water supply, (3) a convenient and suitable place to discharge dirt, and (4) electric power. The method is especially adapted to the Cuyuna district because electric power is so convenient and so abundant. A three-phase 60-cycle 35,000-volt hydraulic-generated current is readily available at reasonable rates at almost any place in the producing area of the district. This is stepped down at the mine to 2,200 volts or any voltage desired.

THE CONCENTRATING ORES.

In certain portions of the district there are ores that can-

not be considered usable unless beneficiated. These ores are called "concentrating ores," and the process of beneficiation is washing. Thus far only non-Bessemer ores have been placed in this class. These ores in their natural state represent a product unfinished by Mother Nature.

Cuyuna ores developed from banded, ferruginous cherts and cherty carbonates by the leaching of the chert and the carbon dioxide. The bands are mixtures of minerals, and the predominating mineral determines the name of the bands, thus some are described as being chert bands, iron-oxide bands, carbonate bands, and so on. The pore spaces in the iron-oxide bands contain some silica in the form of chert, both as a filler and as a binder, and, similarly, the chert bands contain some iron-oxide more or less firmly attached to the grains of chert. In nature the concentration of the ferruginous cherts takes place principally by waters leaching the silica, and the cherty bands generally break down first. In places the chert has been incompletely removed and remains in a disintegrated condition, as a fine powder, which must be removed mechanically before the iron-oxide is usable. It can be removed mechanically by washing the ore with log washers and by jigging.

It is a logical deduction that if natural processes of concentration have been incomplete, then the chert in the iron-oxide bands remains cemented in the pore spaces, and it is this chert which accounts for a siliceous ore after the loose or disintegrated chert has been removed mechanically by washing. This class of iron-ore formation occurs in all parts of an ore deposit, and in some deposits it is very abundant. Sometimes it is very finely banded and sometimes very coarse. In places the banding has been obliterated and the powdery chert and the iron-oxide occur in little pits and as patches. When dry the chert is like a flour, and from some hand specimens can be shaken like fine sand or dust. Again, much of it is so securely attached to the particles of iron-oxide that not even fine crushing will loosen it.

In washing this material it is not alone the removal of the chert that must be considered, but also the character of the ore that results. Washing will remove the external, detached and disintegrated chert. If the ore is first finely ground more chert can be removed but then the fineness of the ore detracts from its value. Ore low in iron, if left coarse, will still be siliceous after washing. If originally low in iron because of the abundant firmly attached chert present, as well

as the loose disintegrated chert, then washing can raise the iron content by only a small percentage at the best. The irregularities in the occurrence of such material in the deposit greatly modify the problem.

The Experiment Station of the School of Mines of the University of Minnesota has just published Bulletin No. 3, which gives preliminary results of a large number of washing tests on small samples of Cuyuna ores. The report shows that in general the material worked resulted in a 25 per cent. loss in material and a 7 per cent. gain in iron units. The Inland Steel Company has just put into operation a small and simple washing plant at its Thompson mine, and the Pittsburgh Steel Ore Company a more elaborate one at its Rowe mine. Even the crudest washing experiments have shown that the percentage of iron can be raised. Nevertheless, the work in this field must still be considered largely experimental. The operators have, it is true, at least made usable for themselves certain material which would otherwise be waste, and perhaps also troublesome. However, it must be borne in mind that the companies now concentrating ores are using the concentrates in their own furnaces, and therefore it would seem that the question is not whether or not Cuyuna ores can be concentrated, but whether a set of circumstances justifies the effort.

THE MANGANIFEROUS IRON ORES.

Probably no feature of the Cuyuna district has been so much heralded abroad as its manganese. Surely no other feature has been so grossly misrepresented and so little understood. The purpose of the following discussion is to present some of the facts.

There seems to have been a certain charm cast about the word manganese, what with war prices for ferro-manganese hovering around a mark like \$105 per ton, and certain Cuyuna properties containing virtually nothing but manganiferous material, the owners or operators of such material are most anxious to find a market for it.

Nearly every North Range iron ore deposit contains manganiferous material, in contrast with the South Range deposits, which contain uniformly less than 1 per cent. of manganese. This material occurs in all positions in the orebodies, in some deposits on the footwall only, in others on the hanging wall only, in others interbedded with the iron ore,

and in others constituting the entire deposit. Some formation samples have averaged over 50 per cent. in manganese. Where the manganese runs high the ore shows a tendency to be nodular; but otherwise the structure is like that of manganese-free iron ore.

Of all the Cuyuna properties that have ever been operated manganiferous ores have been shipped from but three. In 1914 the Iroquois Iron Company shipped about 2,000 tons averaging about 10 per cent. in manganese that had been stockpiled during the winter at their Armour No. 2 mine, now operated by the Inland Steel Company. This manganiferous ore occurred in the midst of their iron-ore orebody. The Iron Mountain Mining Company in 1914 shipped about 600 tons for experimental purposes. This property is now being opened on a larger scale than heretofore and a few cars are being loaded every day as the underground work is extended. Practically the entire deposit is manganiferous material. The Cuyuna-Mille Lacs mine of the American Manganese Manufacturing Company shipped 25,000 tons in 1913, 51,000 tons in 1914 and expects to ship 40,000 tons or more this year. This property, so far as developed and known, contains only manganiferous ores. These ores have been shipped to the Standard Iron Company, Ontario; the Illinois Steel Company, Chicago; the Dunbar Furnaces, Bethlehem Steel Company, Pittsburgh Steel Company and Cambria Steel Company, all of Pennsylvania; the Lake Superior Iron & Chemical Company at Ashland, Wisconsin, and its various plants in Michigan, and probably to other companies also.

Because of the sale these Cuyuna-Mille Lacs ores have found, they may well be considered more fully. The first few carloads shipped averaged from 31 to 33 per cent. in manganese and 25 to 28 per cent. in iron, the combined metallic units being about 58 per cent. It is doubtful, however, if so high a manganese content could be long maintained without depleting the tonnage. The ores of such grade have to be selected carefully and slowly from the larger mass of lower grades, which makes the cost of production high. The operators now offer their ores in four grades, A, B, C, and D, which, as shown in the 1914 yearbook of the Lake Superior Iron Ore Association, analyze as follows:

GRADE	IRON		PHOSPHORUS		MANGANESE		SILICA		MOIS- TURE
	Dried	Natural	Dried	Natural	Dried	Natural	Dried	Natural	
A	37.13	33.417	.108	.0972	23.02	20.718	9.00	8.100	10.00
B	39.77	35.296	.107	.0950	17.44	15.478	10.36	9.195	11.25
C	39.87	35.708	.082	.0734	12.84	11.500	15.80	14.150	10.44
D	40.00	35.80	.071	.0637	10.07	9.033	21.35	19.151	10.30

The basis for this classification is manganese content, the A grade containing 20 to 25 per cent. manganese, the B grade 15 to 20 per cent., the C grade 10 to 15 per cent. and the D grade 10 per cent. and under. The natural metallic content of the A grade is 54.1 per cent.; of the B grade 50.7 per cent.; of the C grade 47.2 per cent., and the D grade 44.8 per cent. The phosphorus is relatively low in all four grades, but especially so in the D grade or low-manganese ores, and the silica increases rapidly with the decrease in the total metallic units.

It is important to state that in most instances the buyer has always specified low phosphorus and low silica, and the largest orders filled have called for the lower-grade mangiferous ore. This necessitates mixing some high-manganese ores with the low-manganese ores in order to keep the silica down. Presumably these low-manganese ores found a sale because they had a low phosphorus content. It therefore resolved into this, that a 10 per cent. manganese ore was salable provided the silica and phosphorus contents were sufficiently low, and to attain the desired percentages of phosphorus and silica the operator had to draw upon an ample quantity of high-manganese ore for mixing purposes. That then would seem to be the "yardstick" with which other prospective operators in mangiferous ores must measure.

There is in the district an almost unlimited quantity of mangiferous ore averaging from 6 per cent. to 20 per cent. in manganese but high in silica and phosphorus, but this material appears to be used in too small quantities to merit consideration here.

There seems to be an opinion current, at least locally, that the European war has caused a much increased demand for American manganese and mangiferous iron ores and

that the ores from such manganiferous deposits as those of the Cuyuna should be in great demand and that the mines should be worked to the very limit. But thus far operations at the Cuyuna manganiferous deposits have been slack. The manganese ores used in the United States are nearly all imported from India, Russia and Brazil, and the present reduction in tonnage from India and Russia is partly overcome by increased imports from Brazil. The manganiferous iron ores used in the United States are largely from the Lake Superior region, and in 1914 amounted to 445,827 tons. According to the reports issued by the United States Geological Survey, only about 15 per cent. of this tonnage, or about 60,000 tons, contained more than 15 per cent. of manganese and this went into low-grade ferro-manganese. This sort of material is rather common in certain localities in each of the various Lake Superior iron-producing districts, and, more important, little is ever sold on the open market. This suggests that operators without furnace affiliations will have to create their own markets for their ores. Again, in 1914, the marketed tonnage of spiegeleisen decreased by 25 per cent. and imports increased from 100 to 3,000 tons, and spiegeleisen is the very product for which Cuyuna manganiferous ores could best be used. This spiegeleisen was largely made at tidewater points in New Jersey. As a result certain manganese deposits in Virginia are now being investigated. It develops then that imported manganiferous material at tidewater points is more desired by furnaces without mines of their own than Lake Superior ores which, by comparison, are low in manganese and which have a long expensive rail haul to the Atlantic border.

This then leads to the question of prices paid for ores involving different unit prices for both iron and manganese and penalties for high silica and phosphorus, all varying widely; but this is a matter outside the scope of this paper. Market prices and market points and consumption have been mentioned only because they are important factors in determining production.

CONCENTRATION OF CUYUNA ORES.

BY EDMUND NEWTON, MINNEAPOLIS, MINN.*

At the present time there are two companies operating iron ore concentration plants on the Cuyuna Range—The Inland Steel Company at the Thompson mine, Crosby, and the Pittsburgh Steel Ore Company at the Rowe mine, Riverton. The former plant began operations in June, 1915; the latter has been running not over a month. Naturally it is too early to describe the practice in these plants since certain changes may be required owing to the many new problems involved.

This paper will be limited to the consideration of the character of the iron ore material which is below shipping grade and certain technical possibilities and limitations involved in any attempt to increase the iron content of the same. The Minnesota School of Mines Experiment Station has been making a general study of various types of low-grade Cuyuna ores with this in view. A number of large samples have been collected and much experimental work done. It is not claimed that these samples represent all the types of low-grade material, but the information obtained from tests already made gives a fair idea of the behavior of ore of this class. The results of the above mentioned work and the conclusions that can be drawn at the present time are herewith presented.

The School of Mines Experiment Station has collected samples of manganiferous iron ores of several types from the Cuyuna Range and is studying the possibilities of beneficiation. The results obtained offer little or no encouragement. No definite statements can be made at the present time.

CHARACTER OF CUYUNA ORES AND THE POSSIBILITIES OF CONCENTRATION.

In considering the application of concentration process to the low-grade iron bearing material, it is essential to be fa-

*Metallurgist of Experiment Station, Minnesota School of Mines.

familiar with the physical and chemical characteristics of the impurity to be removed. Chemical analyses merely indicate the amount of impurity and give no clue to the possibilities of removal.

A clear idea of the origin of the Cuyuna iron ores is of great assistance in explaining many important points. It is not intended, here, to discuss the merits of the geological theories involved nor to offer any new explanation, but merely to present the generally accepted principles and emphasize those principles which have direct bearing on the concentration problems. The original material of the iron formation was probably a cherty iron carbonate. The development of the ore from this material took place in two stages. The cherty iron carbonate was first altered to ferruginous chert by oxidation and hydration of the iron minerals, and second the silica was leached out of the ferruginous chert by the action of alkaline water. The surface waters circulating downward through the formation were responsible for these changes. This leaching of the silica has been more active in the upper portions of the formation and especially along certain channels which allowed of more active flow. Where the leaching of the silica has been nearly complete, the orebodies consist of relatively high grade ore. The low grade material represents various intermediate stages in which leaching has been more or less incomplete. Nearly all gradations from relatively unaltered ferruginous chert to high grade ore may be seen in properties already developed.

The following is a summary of the average chemical composition of the material of the iron formation in the several progressive stages of alteration from cherty iron carbonate to merchantable ore:

TABLE I.

	Cherty Iron Carbonate.	Ferruginous Chert.	Hard Cherty Ores.	Sandy Ore.	Merch. Ore.
Per cent iron (dry)...	27.87	32.76	38.85	47.72	56.77
Per cent. silica (dry)...	38.10	47.49	37.68	22.29	10.01
Per cent. Phos. (dry)....12	.27	.17
Per cent. Alum. (dry)....62	1.53	3.06	3.00
Per cent. Lonl.(dry)...	14.08	3.41	5.56	6.18	7.00

The above table shows the relation between the increase of iron content and the decrease of silica. Silica occurs in these transitional phases as "free" or visible silica and microscopic silica intimately associated with the iron oxide. The occurrence of silica in these two forms and the relative amounts of

each remaining at the several progressive stages of the leaching process is the key to the possibilities of mechanically increasing the iron content.

Ferruginous chert usually consists of silicious hydrated iron oxide, interbanded with hard unaltered chert. The material referred to in Table 1 as "hard cherty ore" is very similar to ferruginous chert with the exception that some silica has already been removed by leaching. Figure 1 is an ideal representation of the banded structure of "hard cherty ore." The entire material carries 38.85 per cent. iron and 37.68 per cent. silica in the dried sample. The ore bands appear to the naked eye to be high grade ore, but contain only 47.00 per cent. iron and 26.25 per cent. silica. The chert bands are relatively

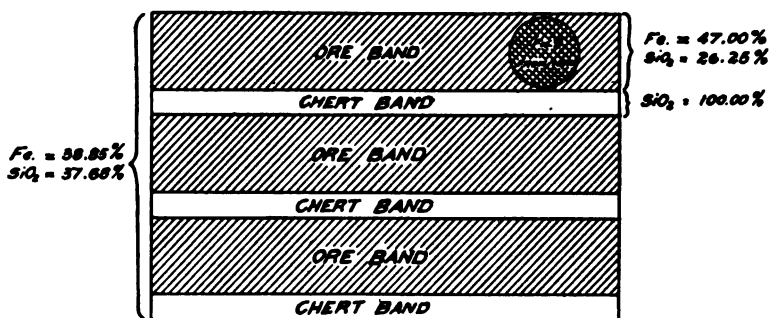


FIG. 1.

IDEAL STRUCTURE OF HARD CHERTY ORE
SHOWING ALTERNATE BANDS OF SILICEOUS IRON OXIDE
AND NEARLY PURE CHERTY SILICA

hard pure cherty silica. These chert bands represent the "free" or visible silica mentioned above. There is 26.25 per cent. silica in the ore bands. It is microscopic or "intimately associated" silica, hence not visible to the naked eye.

The small circle in the upper ore band represents material from which a thin section was made for observation under the microscope. The large illustration, Fig. 2, is a photo-micrograph of this thin section, magnified 45 diameters. The white portions are "intimately associated" silica. The dark portions are iron oxide. It is readily seen that this form of silica is in very intimate mechanical association and quite evenly distributed throughout the groundmass of iron oxide. By comparison with the scale it is readily seen how minute the majority of

these particles are. Practically all are less than 0.01 of an inch in diameter and many are less than 0.001 of an inch.

The further leaching of "hard cherty ore" produces a later transitional stage which we have termed "sandy ore." (See Table 1). The chert bands have become disintegrated from the action of the surface waters, resulting in finely divided powdery silica which is ordinarily spoken of as sand. Some of this has gone into solution and has been removed. The alkaline surface waters, at the same time have apparently dissolved some of the intimately associated silica of the ore bands making this material more porous and naturally higher in iron content.

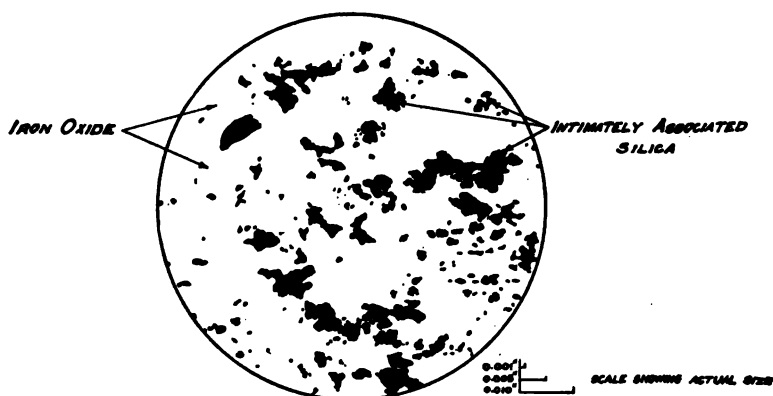


FIG. 2

**PHOTOMICROGRAPH OF ORE-BAND
SHOWING INTIMATELY ASSOCIATED SILICA
NO SILICA VISIBLE TO NAKED EYE
MAGNIFIED 45 DIAM.**

IRON	47.00%
INTIMATELY ASSOCIATED SILICA	26.25%
DRIED AT 212°F.	

Illustration, Fig. 3, is an ideal representation of "sandy ore." It shows the same banded structure as the "hard cherty ore" but with the bands of chert disintegrated to fine sand. The entire material carries 47.72 per cent. iron and 22.29 per cent. silica in the dried sample. The ore bands have increased to 54.83 per cent. iron, with 11.02 per cent. silica.

From the above statements it will readily be seen how directly the physical character of silica affects the possibilities

of concentration. It is evidently impossible to remove intimately associated silica by any economic mechanical process. The visible silica can be removed by log washing or jigging, and the grade of the product produced by these methods will depend upon the amount of intimately associated silica which has been left in the ore bands after leaching. This depends not only upon the amount which has been subsequently removed by the surface waters, but also upon the original condition of sedimentation. We may summarize the conditions which effect the possibilities of concentration as follows:

1. The relative amounts of the two forms of silica which were laid down in the original banded cherty carbonate.

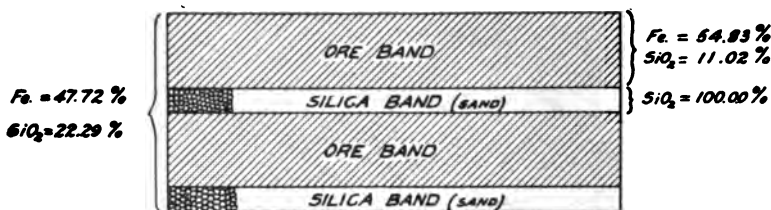


FIG. 3.

IDEAL STRUCTURE OF "SANDY ORE"
SHOWING ALTERNATE BANDS OF SILICEOUS IRON OXIDE
AND BANDS OF NEARLY PURE SILICA SAND.
THIS REPRESENTS A LATER STAGE IN THE LEACHING
OF MATERIAL SIMILAR TO FIGURE 1.

2. The relative amounts of each form of silica which remain after the several stages of the leaching process.

It is generally accepted that the cherty iron carbonate was usually laid down as a banded material consisting, alternately of silicious iron carbonate and cherty silica. Each band representing different conditions of sedimentation. There was no structural change in passing to the ferruginous chert other than the development of pore space. Consequently, the conditions of sedimentation might have been such, that before the leaching of the surface waters began, some ferruginous chert would contain more intimately associated silica in the ore bands than others. A ferruginous chert composed of relatively wide bands would, after leaching, be more readily concentrated mechanically than ferruginous chert of relatively narrow bands.

In order to approximate the relative amounts of each

class of silica which is removed during the leaching processes, five typical samples representing the various stages of leaching have been selected. The amount of silica intimately associated with the ore bands has been computed quantitatively and the results represented by illustration in Fig. 4. It is generally accepted that only silica is removed during the leaching process. The diagram has been constructed accordingly. One hundred units of hard cherty ore, carrying 41.43 per cent. iron and 40.18 per cent. total silica (after correcting

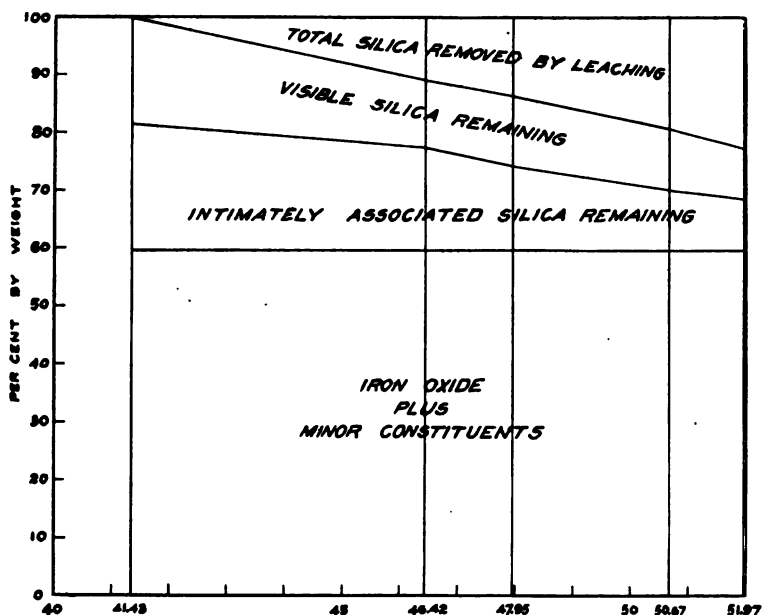


FIG. 4

DIAGRAM SHOWING RELATIVE AMOUNTS OF TWO FORMS OF SILICA REMAINING AT DIFFERENT STAGES OF LEACHING.

for removal of loss on ignition) were used as the basis. The four other samples of ore contained 46.42, 47.95, 50.67, and 51.97 per cent. iron. They represent the material derived by progressive stages of leaching. The upper portion of the curve shows the total amount of silica removed by leaching. With the actual amounts of iron oxide and minor constituents represented on the lower portion of the curve remaining constant, the intervening area indicates the proportional amounts of "free" or visible silica and the intimately associated silica

which remain in the material at any particular stage. It is interesting to note that the intimately associated silica appears to decrease more rapidly than the free or visible silica. Sufficient evidence is not at hand to accept this statement as final.

Results of Actual Tests—The School of Mines Experiment Station has issued a bulletin entitled "Preliminary concentration tests on Cuyuna ores." This bulletin describes in detail the manner in which the tests were made and the full details of the tests. In this paper it does not seem desirable to take up the work in this detailed manner, but rather to include the results of certain typical tests and discuss the general features. The following samples, numbered 1, 2, 3, and 4 were taken from one mine on the North Range. Log washer tests were made on samples 1, 3, and 4; sample 2 was of such character that a jigging test seemed to be desirable.

The Experiment Station log washing plant is similar in design to those in actual operation on the Western Mesabi Range. It consists of an 8-ft. log washer, standard size concentrating tables, and accessory apparatus. It has a crude ore capacity of 30 tons in ten hours. The jigging plant consists of a three-cell Woodbury unit similar to the standard machines of this type. It has a crude ore capacity of approximately one ton per hour. The following are the results on the tests:

ORE NO. 1.

Description—Sandy in appearance. Lumps of relatively low grade dark brown to black hydrated hematite. Some cherty silica adhering to the coarser lumps.

Screen Analysis—

On Mesh.	Per cent. by Wgt.	Per cent. Fe.	Per cent. Iron by Wgt.
1	8.79	37.65	7.62
2	15.51	46.27	16.53
4	22.09	50.31	25.60
10	22.32	51.43	26.45
20	8.82	49.75	10.11
40	4.84	46.50	5.18
80	2.82	42.47	2.76
100	1.35	35.07	1.10
Thru 100	13.46	15.01	4.65
Unsize	100.00	43.41	100.00

Discussion—Approximately 10 per cent. of the total silica is in the form of particles of chert adhering to the coarser

lumps of iron oxide; silica in the form of sand represents approximately 35 per cent. and silica intimately associated with the particles of iron oxide, 55 per cent. Judging from the screen analysis and an examination of the physical characteristics, log washing would produce a concentrate carrying not over 50 per cent. iron. Jigging would give substantially the same results.

Log Washer Test; Summary of Results—

	Crude Ore.	Concentrate.	Tailing.
Per cent. by weight	100.00	83.38	16.62
Weight in pounds	16,592.72	13,838.58	2,754.14
Per cent. iron	43.41	49.07	14.94
Per cent. silica	29.17	19.73	76.57
Per cent. phosphorus239	.273	.067
Per cent. ignition loss.....	7.28	8.04	3.43
Per cent. iron by weight.....	100.00	94.28	5.72

*Discussion of Results—*The crude ore was raised from 43.41 per cent. iron to 49.07 per cent. iron in the concentrate, an increase of 5.66 per cent. iron. The silica decreased from 29.17 per cent. in the crude ore to 19.73 per cent. in the concentrate, a decrease of 9.44 per cent. Only 43.57 per cent. of the silica in the crude ore was in such form that it could be removed by log washing. The iron recovery was high, showing that the work of the log washer was efficient from the standpoint of saving the iron oxide. Only one pound of metallic iron was removed with 5.12 pounds of silica in the tailing. The tailing carried 14.94 per cent. iron. The log washer concentrate was allowed to drain on a concrete floor for approximately twenty hours. A moisture sample taken at the end of this period showed approximately 8 per cent. moisture. The percentage of natural iron in the concentrate on this basis was 45.14. This is 6.36 per cent. below the non-bessemer base grade. Unless concentrate of this character could be mixed with sufficient high-grade material, beneficiation by log washing would not be commercially successful.

ORE No. 2.

*Description—*Cherty material. The chert does not occur in bands but is largely attached to the coarse particles of iron oxide. There is also a small amount of relatively coarse particles of pure chert. There is very little fine sandy silica.

Screen Analysis—

On Mesh.	Per cent. by Wgt.	Per cent. Fe.	Per cent. Iron by Wgt.
1	13.77	45.26	13.98
2	31.50	46.94	33.18
4	19.69	48.74	21.54
10	16.29	47.06	17.21
20	6.37	44.26	6.33
40	3.58	38.99	3.14
80	2.29	32.61	1.68
100	.94	27.11	.56
Thru 100	5.57	19.05	2.38
Unslized	100.00	44.60	100.00

Discussion—Approximately 42 per cent. of the total silica is in the form of coarse particles of chert; approximately 8 per cent. is "sandy silica" and 50 per cent. intimately associated with the iron oxide particles. Judging from the screen analysis and inspection of the material log washing would not appreciably raise the grade. Crushing the coarse material to pass a $\frac{1}{2}$ -in. screen followed by jigging would probably give better results.

Jigging Test; Summary of Results—

	Crude Ore.	Combined Concentrates.	Combined Tailings.
Per cent. by weight.....	100.00	67.87	32.13
Weight in pounds	14,791.32	10,039.49	4,751.83
Per cent. iron	44.60	51.54	29.93
Per cent. silica	32.59	16.63	47.64
Per cent. phosphorus309	.361	.201
Per cent. ignition loss	6.99	7.80	5.27
Per cent. iron by weight.....	100.00	78.44	21.56

Discussion of Results—The entire crude ore was raised from 44.60 per cent. iron to 51.54 per cent. iron, an increase of 6.94 per cent. iron. The silica decreased from 32.59 per cent. in the crude ore to 16.63 per cent. in the concentrates, a decrease of 15.96 per cent. silica. The concentrates recovery was 67.87 per cent. The iron recovery was 78.44 per cent. Only 57.41 per cent. of the silica was in such form that it was eliminated by jigging. One pound of iron was eliminated with 1.59 pounds of silica in the tailings. The tailings carried 29.93 per cent. iron. With 8 per cent. moisture in the concentrates, the natural iron content is 47.42 per cent. or 4.08 per cent. below the non-bessemer base grade.

ORE No. 3.

Description—Sandy in appearance. The lumps are of rel-

atively high-grade brownish-black hydrated hematite with very little visible cherty silica attached.

Screen Analysis—

On Mesh.	Per cent. by Wgt.	Per cent. Fe.	Per cent. Iron by Wgt.
1	3.74	48.73	3.86
2	11.46	52.32	12.71
4	22.23	53.55	25.22
10	24.10	54.67	27.90
20	10.92	54.00	12.49
40	5.34	52.21	5.91
80	3.27	49.86	3.46
100	1.43	46.05	1.39
Thru 100	17.51	19.04	7.06
Unsize	100.00	47.21	100.00

Discussion—In this ore 77.79 per cent. is coarser than 40-mesh and carries 53.45 per cent. iron. The material finer than 100-mesh amounts to 17.51 per cent. and carries 19.04 per cent. iron. Judging from the screen analysis, log washing would produce a concentrate carrying not over 54 per cent. iron. Jigging would give substantially the same results.

Log Washer Test; Summary of Results—

	Crude Ore.	Concentrate.	Tailing.
Per cent. by weight	100.00	80.13	19.87
Weight in pounds	15,101.35	12,105.20	2,996.15
Per cent. iron	47.21	53.90	20.13
Per cent. silica	24.22	15.24	60.48
Per cent. phosphorus191	.211	.111
Per cent. ignition loss	6.82	7.50	4.08
Per cent. by weight	100.00	91.55	8.45

Discussion of Results—The crude ore was raised from 47.21 per cent. iron to 53.90 per cent. iron in the concentrate, an increase of 6.69 per cent. iron. The silica decreased from 24.22 per cent. to 15.24 per cent. in the concentrate, a decrease of 8.98 per cent. Only 49.55 per cent. of the silica in the crude ore was in such form that it could be removed by log washing. One pound of iron was removed with 3.01 pounds of silica in the tailing. The tailing carried 20.13 per cent. iron. The recoveries are relatively high.

The moisture in the concentrate was approximately 8 per cent. The natural iron content of the concentrate on this basis was 49.59 per cent. or 1.91 per cent. below the non-bessemer base grade.

ORE No. 4.

Description—Sandy in appearance. Similar to Ore 3 except that the coarse lumps are higher in iron content,

Screen Analysis—

On Mesh.	Per cent. by Wgt.	Per cent. Fe.	Per cent. Iron by Wgt.
1	6.66	48.85	6.82
2	14.20	54.67	16.27
4	24.34	55.91	28.51
10	21.38	55.12	24.70
20	8.72	54.22	9.91
40	5.02	51.20	5.39
80	3.48	45.93	3.35
100	1.23	36.41	.94
Thru 100	14.97	13.10	4.11
Unsize	100.00	47.72	100.00

Discussion—In this ore 80.32 per cent. is coarser than 40-mesh and carries 54.42 per cent. iron. The material finer than 100-mesh amounts to 14.97 per cent. and carries 13.10 per cent. iron. Judging from the screen analysis, log washing would produce a concentrate carrying not over 55 per cent. iron. Jigging would give substantially the same results.

Log Washer Test; Summary of Results—

	Crude Ore.	Concentrate.	Tailing.
Per cent. by weight.....	100.00	79.21	20.79
Weight in pounds	16,760.26	13,275.98	3,484.28
Per cent. iron	47.72	54.83	20.60
Per cent. silica	22.29	11.02	65.26
Per cent. phosphorus269	.321	.070
Per cent. ignition loss	8.18	9.20	4.26
Per cent. iron by weight.....	100.00	91.02	8.98

Discussion of Results—The crude ore was raised from 47.72 per cent. iron to 54.83 per cent. iron in the concentrate an increase of 7.11 per cent. iron. The silica decreased from 22.29 per cent. in the crude ore to 11.02 per cent. in the concentrate, a decrease of 11.87 per cent. silica. In this ore 60.85 per cent. of the silica was in such form that it could be eliminated by log washing. One pound of metallic iron was removed with 3.16 pounds of silica in the tailing. The tailing carried 20.60 per cent. iron. The recoveries are relatively high.

The moisture in the concentrates was 8 per cent. making the natural iron content of the concentrate 50.44 per cent. iron or 1.06 per cent. below the non-bessemer base grade.

Behavior of Constituents—Results of the tests made on these samples indicate the behavior of the various constituents during the processes of concentration. The following tables show the percentage increase or decrease of iron, phosphorus, ignition loss, and silica from crude ore to concentrates:

TABLE NO. 2.

Ore No.	Per cent. Fe. Crude.	Per cent. Fe Concentrates.	Ratio of Increase.
1	43.41	49.07	1.1304
2	44.60	51.54	1.1556
3	47.21	53.90	1.1417
4	47.72	54.83	1.1489
Average	45.74	52.33	1.1441

Phosphorus—

Ore No.	Phos. Crude.	Phos. Concentrates.	Ratio of Increase.
1	.239	.273	1.1423
2	.309	.361	1.1683
3	.191	.211	1.1047
4	.269	.321	1.1933
Average	.252	.291	1.1548

Ignition Loss—

Ore No.	Ign. Crude.	Ign. Concentrates.	Ratio of Increase.
1	7.28	8.04	1.1044
2	6.99	7.80	1.1159
3	6.82	7.50	1.0997
4	8.18	9.20	1.1247
Average	7.32	8.14	1.1120

Silica—

Ore No.	Sil. Crude.	Sil. Concentrates.	Ratio of Decrease.
1	29.17	19.73	.6764
2	26.59	16.63	.6254
3	24.22	15.24	.6292
4	22.29	11.02	.4944
Average	25.57	15.65	.6120

Behavior of Iron—During the beneficiation processes the iron analyses increase slightly from crude ore to concentrates. Crude ores high in iron increase by a somewhat greater ratio than those lower in iron.

Behavior of Phosphorus—The above summary shows that phosphorus increases from crude ore to concentrates in about the same ratio as the iron. This indicates a very intimate association between the phosphorus and iron. Elimination of phosphorus by any method of beneficiation would probably be difficult.

Behavior of Ignition Loss—The above summary shows that the ignition loss increases from crude ore to concentrates in a slightly smaller ratio than the iron.

Behavior of Silica—Silica decreases during the beneficiation processes.

SUMMARY.

The following conclusions have been reached from the preliminary study of the Cuyuna ores:

(1) The physical character of the low-grade iron-bearing material is exceedingly variable. Some material may be treated by log washing, some may require jigging, but as a rule, only few ores can be beneficiated by either process.

(2) The grade of concentrates produced is usually proportional to the grade of crude ore. The grade of product is dependent upon the iron content of the original bands of iron oxide. These contain more or less intimately associated silica which cannot be eliminated by log washing or jigging. The amount of such silica depends upon the extent of the natural leaching. Consequently, very low-grade crude ores yield a low-grade product, while higher-grade crude ores will yield a higher-grade concentrate. This does not always hold true of Mesabi Range ores.

The average grade of samples from the Cuyuna Range which have been tested by the Experiment Station is 45.93 per cent. iron. The average concentrates produced from these samples by log washing and jigging represents 75.24 per cent. of the original crude ore and carries 51.65 per cent. iron. This is an increase in iron content of less than 7 per cent. Later work has developed ores better suited to beneficiation than those previously received by the Experiment Station.

The average grade of twenty-seven samples of material from the Western Mesabi Range which have been tested by the Experiment Station is 44.85 per cent. iron. The average concentrates produced from these samples by log washing represents 62.74 per cent. of the original crude ore and carries 56.28 per cent. iron. This is an increase in iron content of approximately 12 per cent.

(3) Table treatment of log washer tailing is not successful. The log washer tailing is generally low in iron content and much of the iron is in the form of a colloidal slime. Tables produce a very small amount of concentrate carrying a relatively low percentage of iron.

(4) Phosphorus increases from crude ore to concentrate. The tests made show that phosphorus is concentrated in about the same ratio as iron. This indicates an intimate association between the two elements.

PROGRESS IN UNDERGROUND ORE LOADING.

BY M. E. RICHARDS, CRYSTAL FALLS, MICHIGAN.*

The desire to lower mining costs, coupled of late years in some cases with a scarcity of labor, has fixed the attention of mining men upon mining machinery as one of the important means to greater efficiency. Notable advancement has already been made in this field, and the needs of the day are keeping mining men constantly on the alert for still greater progress. Every machine used in mining has been improved, and a special effort has been made to substitute power operation for hand labor. This is particularly true of drilling in mines. The efficiency of drilling machines has been increased so that a greatly increased amount of ground can be broken by one man in an hour; but there is still a great amount of time and money spent in shoveling or mucking, especially at the present time in connection with drifting and tunnelling operations. After the rock and ore is broken, with but a few exceptions, mine operators are still using ancient methods of loading ore by hand methods, which were used over two thousand years ago. Many attempts have been made to eliminate this mucking by hand, which incidently is the hardest manual labor underground, and several machines have been tried for this purpose; nearly all of them have failed, however.

The efficiency engineers are at present making time studies of the operations in mucking; there have been careful investigations of late of the advantages of long- and short-handled shovels; a relay shoveling system has been worked out and applied; car bodies have been lowered to reduce the effort of loading and all of these activities and improvements are bound to produce results. However, after all, it is the mechanical shovel to which we must look for the final solution of the problem. When this comes, the system which has seen no

*General Manager, Judson Mining Co.

change since mining began, if not completely revolutionized, will at least be greatly improved.

To be successful a mechanical shovel must satisfy the following requirements:

The first cost must be reasonable; that is, it must be low enough so that there shall be no doubt that the investment will yield a good return. This also means that the machine must have sufficient capacity to do the work of several muckers.

The machine must be simple, durable and not liable to break down, and all parts must be readily accessible for removal, adjustment and replacement. It must be a machine which can be handled by miners or handy men, easily guided and controlled, and which is as near fool-proof as possible.

It must be able to handle sticky ore, wet ore, dry ore, chunks weighing from 60 to 100 pounds, and even the larger pieces occasionally encountered. All machinery and working parts must be completely housed and protected from dirt and water.

The motion of the machine must be such that, if a wall or big chunk of ore or other firm objects are struck, the mechanism will not be damaged. It must be able to shovel, convey and dump broken materials.

It must be so designed as to permit of its use in drifts 6 ft. by 6 ft., and at the same time so that it can be taken down through a shaft and through openings considerably smaller. It must also be easy to move to and from the breast of the drift, on its own power if necessary.

To combine all of the above in one machine is no doubt difficult, but actual observation of the several machines now on the market indicates that it has been already accomplished.

Realizing the need for a mechanical shovel that will work underground, urged on by predictions of a labor shortage, individuals in almost every mine organization set their minds to work and evolved ideas for, and in many cases actually set to working out, a practical machine. Such activity has not been confined to recent years, for I find on investigation that already about twenty years ago, a conveyor-type loader was tried out at the Fayal mine of the Minnesota Iron Mining Company.

About ten years ago Thompson & Greer tried out a machine of their own design of the conveyor-type at the New-

port property. The machine showed some saving in operations, but was out of order a large part of the time.

About four years ago Mr. Nels Flodin of Marquette, started with a drag shovel, the special feature of which was a small drum which pulled an ordinary drag scraper, fastened to a rope, up an incline and dumped it automatically into a car. The scraper had to be pulled back by hand; it was this feature of its operation which probably caused the machine to be abandoned.

During the present year The Cleveland-Cliffs Iron Com-



PORTABLE LOADER AT THE NORTH LAKE MINE OF THE CLEVELAND-CLIFFS
IRON CO., ISHPeming, MICH.

pany has worked out and is using a conveyor-type loader at the North Lake property. It is considered a success and has demonstrated itself to be a labor saver and it has considerably increased the speed in drifting. The output has been increased from 25 to 30 per cent. The arrangement is simple: Men shovel by hand onto the belt of a belt-conveyor, which runs up an incline and discharges into a tram car. (See cut).

During the present year there has been a loading machine in daily operation in the Morton mine on the Mesaba Range,

and it has been giving very satisfactory results. It was designed by Billings and Middlemiss of the Morton mine. This machine consists in general of a large hoe which reaches out and drags ore onto an apron, which in turn discharges it onto a belt-conveyor which elevates it into the ore car. The original idea was to adopt the three motions of a steam shovel to a machine for drift use, and this machine has been built around that idea. It consists essentially of three air cylinders; one operating the in-and-out motion of the hoe; one the swinging



BILLINGS AND MIDDLEMISS SHOVELING MACHINE

motion, and the third the tilting motion. A reciprocating air engine drives the conveyor and propelling mechanism. One man can run the loader and operate the hoe; another swings and tilts the conveyor and propelling mechanism from the operator's stand. Some very good results in cleaning up a breast of thirty tons of ore have been obtained, the time required in wet dirt averaging two hours; in dry places this amount of ore has been taken out in an hour. (See cut).

Another type of loading machine has been worked out by Mr. Sam Hoar of Virginia, Minnesota. Considerable experimental work is being done with this machine, but to my knowledge it has not yet been tried out in practical service. This machine employs an air cylinder which runs a shovel out on a beam, takes a load and drags it back onto the conveyor, which discharges into an ore car.

The McDermott machine is of another type. This, too, is still being worked on at the present time, and to my knowledge is not at present in actual service. This machine as originally conceived, as I understand it, was built around the steam shovel idea with many modifications to fit underground conditions.

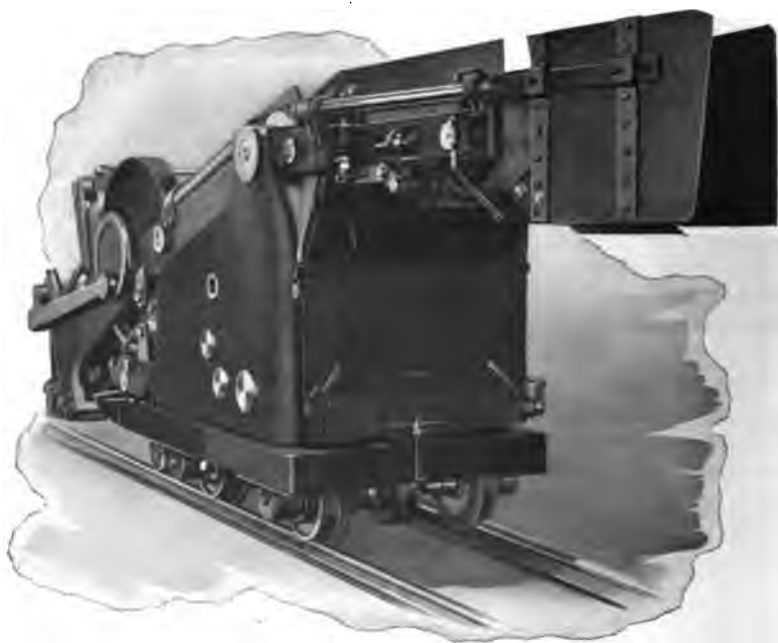
There are two machines on the market today which are being used in actual mining work. The older of the two is the Meyers-Whaley machine, which is manufactured and used considerably in the South. This machine has been doing very efficient work, cutting down loading costs to a great extent, and is a great labor saver. On the front end of this machine there is a shovel with automatic cam motion, which discharges the ore onto the bottom of the belt conveyor which elevates the ore and discharges it in the rear into a car. The power used for this machine is either electric or compressed air. I understand that this machine is approximately 24 ft. long and weighs about $8\frac{1}{2}$ tons. Its length would probably prevent its general use in the Lake Superior iron ore mines, as it could not be readily moved around our sharp curves and small openings without being dismantled. There is no doubt, however, that under conditions where this machine can be used, it will greatly reduce the cost of drifting and mucking; in fact, this has been proved in the case of the many machines which have been turned out by the Meyers-Whaley Company.

The latest development in a mechanical shovel is the machine known as the Halby shoveling machine, manufactured by the Lake Shore Engine Works of Marquette, Michigan. This should be of interest to members of the Lake Superior Mining Institute because it has been developed in the Lake Superior district and is designed primarily for use in its mines. This machine was first conceived about three years ago, and the first completed machine was shown at our meeting on the Marquette Range last year (1914). (See cut).

The machine as marketed today has an overall length of 15 ft., but it can in a very short time be shortened to an

overall length of 10 ft., if conditions necessitate the shorter length. It can be arranged to any gauge from 18 in. up to 44, and is designed for operation on curves of 25 and possibly 20 ft. radius. The overall height of the machine is 5 ft. 4 in., and the total width approximately 4 feet. Its total weight is nine thousand pounds.

The machine is arranged for air, gasoline, or electric operation, and requires one man to run it. It is made up in



REAR VIEW—HALBY SHOVELING MACHINE

three distinct sections, each of which forms a unit by itself; this permits the machine to be taken down very small shafts. The top or conveyor section contains the working parts for the conveyor and shovel mechanism. The center or power section contains the motor power, clutches and gear drives. The lower or truck section is the traveling support of the entire machine.

The motion of the Halby shoveling machine corresponds very closely to the motion of a hand mucker with a shovel. A shovel 22 in. wide is actuated by a lever, which gives it a forward movement when at the bottom of its travel. The

machine itself in being propelled forward furnishes the power which thrusts the shovel into the pile. To withdraw the shovel, the machine is moved backward; but this is unnecessary, for the shovel will lift itself. When lifted, it takes an incline position at an angle of about 65° , delivering the load onto the conveyor belt which is running continuously and which carries the ore over the machine and back into the ore car. An automatic deflector delivers the ore into the center of the car, irrespective of the position of the shovel in the face of the drift.

One man operates the machine by means of levers on a platform placed at the side of the machine. The dipper runs



HALBY SHOVELING MACHINE—SHOWING DUMPING POSITION OF SHOVEL

automatically, thrusting into a bank of ore, raising the load up and discharging out of the back of the dipper onto the bottom end of the conveyor-belt, as described above. The shovel runner merely controls the propelling motion of the machine to keep it well up against the bank or swings the shovel from one side of the drift to the other as the pile is loaded, or raises or lowers the beam which holds the dipper, so that it will dig at any elevation required. There are four levers on the platform, two of which are used continuously.

The Halby shoveling machine has been tried out during the present year at the Judson mine on the Menominee Range, and the results have been very satisfactory. The cost of drifting has been reduced to a considerable extent, and it has been found that one man with this machine can do, in the same time, the work which it takes 12 men with shovels to do. This machine is at present loading a 35 cu. ft. car (two tons), in an 8- by 8-ft. drift, in $1\frac{1}{2}$ min. when loading conditions are fair, that is, when there is a fair size bank of ore in front of the machine. At times the ore bank is smaller and scattered, and under such conditions it takes as high as 4 min. to load a two-ton car. Formerly with hand labor it required 2



SIDE VIEW—HALBY SHOVELING MACHINE

men shoveling 20 min. to load a two-ton car under the same conditions that the machine is now working.

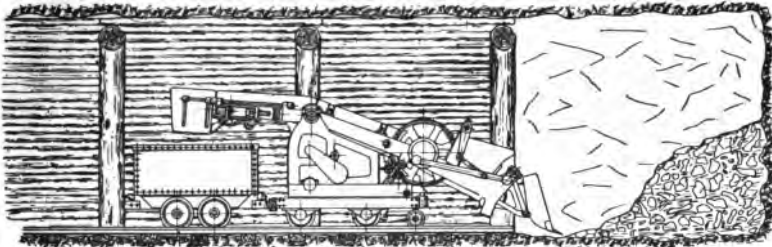
The cost of operating the machine an 8-hour shift has been estimated as follows:

Power	\$3.00
Runner	2.50
Interest on investment40
Repairs50
Oil, etc.17
Total	\$6.57

Figuring 200 tons loaded per 8-hour shift, this is a net cost per ton of .032c with the machine. The labor expense of two men loading by hand for an 8-hour shift would be

\$5.10. Under similar conditions they would load approximately 30 tons in 8 hours at a cost of 17c per ton.

The experience at the Judson mine has very clearly demonstrated that a mechanical shovel will not only considerably reduce the mucking cost, but will also enable a much more rapid advancement in drifting. I do not desire to give the impression that a mechanical shovel can be used to advantage in mining under all conditions, for this would not be true. These machines would not reduce the cost where the ore is milled down from stopes, and it is not necessary to shovel, as in back-, sub- and block-stoping; but in these mines it can be used to great advantage in development work, by reducing the cost per foot in drifting and by speeding up the development work. A mechanical shoveling machine can be worked to advantage in practically every place where the hand shovel is used in drifting, trenching and developing, and in the slicing and caving system of mining, and in all classes of mining where it is necessary that the ore be shoveled, and in all openings in which the machine will operate.



SHOWING PROPORTIONS—HALBY SHOVELING MACHINE IN 8x8 FT. DRIFT

PAST OFFICERS.

PRESIDENTS.

Nelson P. Hulst1893	
J. Parke Channing1894	James MacNaughton1905
John Duncan1895	Thomas F. Cole1906
William G. Mather1896	Murray M. Duncan1908
William Kelly1898	D. E. Sutherland1909
Graham Pope1900	William J. Richards1910
W. J. Olcott1901	F. W. Denton1911
Walter Fitch1902	Pentecost Mitchell1912
George H. Abeel1903	W. H. Johnston1913
O. C. Davidson1904	L. M. Hardenburgh1914

(No meetings were held in 1897, 1899 and 1907).

VICE PRESIDENTS.

1893.		
John T. Jones		Graham Pope
F. P. Mills	J. Parke Channing	M. W. Burt
1894.		
John T. Jones		Graham Pope
F. P. Mills	R. A. Parker	W. J. Olcott
1895.		
F. McM. Stanton		Per Larsson
Geo. A. Newett	R. A. Parker	W. J. Olcott
1896.		
F. McM. Stanton		Per Larsson
Geo. A. Newett	J. F. Armstrong	Geo. H. Abeel
1898.		
E. F. Brown		Walter Fitch
James B. Cooper	Ed. Ball	Geo. H. Abeel
1900.		
O. C. Davidson		J. H. McLean
T. F. Cole	M. M. Duncan	F. W. Denton
1901.		
J. H. McLean		F. W. Denton
M. M. Duncan	Nelson P. Hulst	William Kelly

1902.		
William Kelly		H. F. Ellard
Nelson P. Hulst	Fred Smith	Wm. H. Johnston
1903.		
H. F. Ellard		Wm. H. Johnston
Fred Smith	James B. Cooper	John H. McLean
1904.		
H. F. Ellard		John H. McLean
Wm. H. Johnston	Fred Smith	James B. Cooper
1905.		
M. M. Duncan		John H. McLean
Fred M. Prescott	F. W. McNair	James B. Cooper
1906.		
M. M. Duncan		F. W. McNair
J. M. Longyear	Fred M. Prescott	F. W. Denton
1908.		
J. M. Longyear		D. E. Sutherland
F. W. Denton	David T. Morgan	Norman W. Haire
1909.		
W. J. Richards		D. E. Sutherland
Charles Trezona-	D. T. Morgan	Norman W. Haire
1910.		
W. J. Richards		Charles Trezona
John M. Bush	Frederick W. Sperr	James H. Rough
1911.		
E. D. Brigham		C. H. Munger
John M. Bush	Frederick W. Sperr	James H. Rough
1912.		
E. D. Brigham		C. H. Munger
Geo. H. Abeel	W. P. Chinn	W. H. Jobe
1913.		
Geo. H. Abeel		A. D. Edwards
Francis J. Webb	W. P. Chinn	W. H. Jobe
1914.		
Francis J. Webb		A. D. Edwards
Charles T. Kruse	Luther C. Brewer	Charles E. Laurence
MANAGERS.		
1893.		
John Duncan		James MacNaughton
Walter Fitch	William Kelly	Charles Munger
1894.		
Walter Fitch		C. M. Boss
John Duncan	M. E. Wadsworth	O. C. Davidson

PAST OFFICERS

F. P. Mills Ed Ball	1895. M. E. Wadsworth	C. M. Boss O. C. Davidson
F. P. Mills Ed. Ball	1896. C. H. Munger	Graham Pope William Kelly
M. M. Duncan J. D. Gilchrist	1898. T. F. Cole	Graham Pope O. C. Davidson
E. F. Brown Ed. Ball	1900. James B. Cooper	Walter Fitch George H. Abeel
James B. Cooper James MacNaughton	1901. (One Vacancy)	James Clancey J. L. Greatsinger
James Clancey J. L. Greatsinger	1902. Amos Shephard	Graham Pope T. F. Cole
Graham Pope Amos Shephard	1903. W. J. Richards	T. F. Cole John McDowell
John McDowell Wm. J. Richards	1904. Graham Pope	Thomas F. Cole Amos Shephard
John C. Greenway John McDowell	1905. William Kelly	H. B. Sturtevant Wm. J. Richards
John C. Greenway Jas. R. Thompson	1906. William Kelly	H. B. Sturtevant Felix A. Vogel
James R. Thompson Felix A. Vogel	1908. John C. Greenway	J. Ward Amberg Pentecost Mitchell
F. E. Keese W. J. Uren	1909. L. M. Hardenburg	J. Ward Amberg Pentecost Mitchell
Frank E. Keese Charles E. Lawrence	1910. William J. Uren	L. M. Hardenburg William J. West
Charles E. Lawrence Peter W. Pascoe	1911. J. B. Cooper	William J. West L. C. Brewer
M. H. Godfrey Peter W. Pascoe	1912. J. B. Cooper	J. E. Jopling L. C. Brewer

1913.

M. H. Godfrey
G. S. Barber

Wm. H. Johnston

J. E. Jopling
C. H. Baxter

1914.

G. S. Barber
W. A. Siebenthal

*Stuart R. Elliott

C. H. Baxter
J. S. Lutes

*To fill vacancy of Wm. H. Johnston, elected to presidency.

TREASURERS.

C. M. Boss	1893
A. C. Lane	1894
Geo. D. Swift	1895-1896
A. J. Yungbluth	1898-1900
Geo. H. Abeel	1901-1902
E. W. Hopkins	1903-....

SECRETARIES.

F. W. Denton	1893-1896
F. W. Denton and F. W. Sperr	1898
F. W. Sperr	1900
A. J. Yungbluth	1901-....

LIST OF PUBLICATIONS RECEIVED BY THE INSTITUTE.

American Institute of Mining Engineers, 29 West 39th Street, New York City.

Mining and Metallurgical Society of America, 505 Pearl Street, New York City.

American Society of Civil Engineers, 220 West 57th Street, New York City.

Massachusetts Institute of Technology, Boston, Mass.

Western Society of Engineers, 1734-41 Monadnock Block, Chicago.

The Mining Society of Nova Scotia, Halifax, N. S.

Canadian Mining Institute, Rooms 3 and 4, Windsor Hotel, Montreal.

Canadian Society of Civil Engineers, Montreal.

Institute of Mining Engineers, Neville Hall, Newcastle Upon-Tyne, England.

North of England Institute of Mining and Mechanical Engineers, Newcastle-Upon-Tyne, England.

Chemical, Metallurgical and Mining Society of South Africa, Johannesburg, S. A.

American Mining Congress, Rumsey Bldg., Washington, D. C.

State Bureau of Mines, Colorado, Denver, Colo.

Reports of the United States Geological Survey, Washington, D. C.

Geological Survey of Ohio State University, Columbus, O.

Geological Survey of New South Wales, Sydney, N. S. W.

Oklahoma Geological Survey, Norman, Okla.

University of Oregon, Library, Eugene, Oregon,

Case School of Applied Science, Department of Mining & Metallurgy, Cleveland, Ohio.

University of Illinois, Exchange Department, Urbana, Ills.

University of Missouri, Columbia, Mo.

University of Michigan, Ann Arbor, Mich.

University of Colorado, Boulder, Colo.

Columbia University, New York City, N. Y.

University of Pittsburg, State Hall, Pittsburg, Pa.

Iowa State College, Ames, Iowa.

Iron Age, 239 West 39th Street, New York.

Engineering & Mining Journal, 10th Avenue and 36th Street, New York.

Engineering Magazine, 140 Nassau Street, New York.

The Mining Magazine, 724 Salisbury House, London, E. C.

Mines and Mining, 1824 Curtis Street, Denver, Colo.

Engineering-Contracting, 355 Dearborn Street, Chicago, Ills.

Mining & Engineering World, Monadnock Block, Chicago, Ills.

Mining Science, Denver Colo.

Mining & Scientific Press, 420 Market St., San Francisco, Cal.

The Mexican Mining Journal, Mexico City, Mexico.

Stahl und Eisen, Dusseldorf, Germany, Jacobistrasse 5.

The Excavating Engineer, 267 National Avenue, Milwaukee, Wis.

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